182

JAN 13 1933

Engineering and Mining Journal

WITH WHICH IS CONSOLIDATED *ENGINEERING & MINING WORLD*

NEW YORK LONDON

A McGraw-Hill Publication January, 1933 Price 50 Cents, or 2s. 6d. Per Copy

Cut Your Rock Handling Costs in 1933...

Progressive companies now do it with Eimco-Finlay Loaders .... for example:

North Lily Mining Co., Eureka, Utah—Loading cost reduced to 30% of previous hand loading costs; cost of loader repaid within 2 months.

U. S. Smelting & Refining Co., Lark, Utah—Loading cost cut from 36 cts. per ton to 22.5; loader paid for itself in less than five months (operating view herewith).

Idaho Maryland Mines Co., Grass Valley, Cal.—The most active gold mining company in California, uses three Eimco-Finlay loaders to secure record low costs.

Itogon Mining Co., Baguio, P. I.—Purchased its second loader to further increase gold production at reduced costs.

Alaska Juneau Gold Mining Co., Juneau, Alaska—Purchased a second loader a few weeks after installing the first because "speed in drifting was increased 50%, and rock-loading efficiency increased 100%."

Four Square Gold Syndicate, Wallace, Idaho—Purchased loader to drive 600 feet of 8½ ft. x 9½ ft. drift — "Using your loader we have averaged better than 10 ft. per day with 4 miners; any company figuring on doing any tunneling should certainly have your loader," says its General Manager.

Eastern Iron & Metal Company, Inc.

640 South Fourth West Street, Salt Lake City, Utah, U.S.A.

THESE and other alert companies will build up bigger profits in 1933 by cutting rock-handling costs. You have the same opportunity. Write today. You can buy and pay for one or more Eimco-Finlay Loaders out of savings.

PAYS FOR ITSELF OUT OF SAVINGS

And Now Another Advance In Finer Reduction Crushing

Throughout the World, Symons Cone Crushers are regular equipment in the crushing plants of the larger metal mining companies. Revolutionary as was the Standard Cone for reduction crushing, equally so is the new Short Head Fine Reduction Cone Crusher in producing a still finer product for mill feed.

This new crusher operates on the same principle as does the Standard Cone and utilizes the same process of crushing. It has the additional advantages of increased capacity in the smaller sizes, delivers a more uniformly fine product and enables a higher recovery being secured. Since it produces in one operation a finer mill feed, the overall crushing and ultimate milling costs are materially reduced. Again another outstanding advance has been made in fine reduction crushing.

Let us send you further particulars on this new crusher, and how it can be used to advantage in your crushing operations.

NORDBERG MFG. CO., MILWAUKEE, WIS. U.S.A.

NEW YORK	LONDON, WC 2, ENG.	LOS ANGELES
60 East 42nd St.	Bush House	Subway Terminal Bldg.

SYMONS CONE CRUSHERS

Engineering and Mining Journal, January, 1933. Vol. 134, No. 1. Published monthly by the McGraw-Hill Publishing Company, Inc., 330 West 42d St., New York City, N. Y., U. S. A. Subscription rates, United States, $3 per annum; Canada and other countries in the Americas outside the United States, $4 per annum; elsewhere $5 or 25 shillings per annum. Price per copy, 50 cents or 2s. 6d. Entered as second-class matter June 20, 1879, at the Post Office, New York, N. Y., under the act of March 3, 1879. Printed in U. S. A.

Engineering and Mining Journal

with which is consolidated

Engineering & Mining World

January, 1933 Vol. 134—No. 1

EDITORIAL

THE leading article in this issue, describing the design and constructional details of a large crushing and screening plant recently put into commission on the banks of the Hudson River, near New York City, demonstrates the continuous opportunity that exists for refinements in operating technique. During a period when a lull is occurring in the erection of large new reduction plants for the handling of metalliferous rock, engineers and operators will appreciate an opportunity to learn what important decisions are being made by the executives responsible for an entirely new commercial venture, costing $1,500,000, and designed for an output of 1,000 tons per hour. The article, and the working drawings that add so much to its practical value, disclose engineering features of help and interest to superintendents and designers of other large plants for the reduction of rock and the screening and mechanical treatment of intermediate products. While the article was being set in type a request came from a tin-mining company for information on clay-removal practice that is now to be found in the description of the washing units at the Hudson River plant.

One comparatively heavy expense that must be met by the owners of many industrial units today—whether machinery is idle, operating on part time or on full time —is for fire insurance. The Hudson River company has made and will make no payment of this character. The plant is constructed of concrete, steel, and galvanized copper-steel siding, and no fire hazard exists.

Turning to another opportunity for economy: The largest expense for crushing is for power, and wide interest will be created by the description, following the leading article, of the diesel-electric generating plant, which permits the company to operate independently of a high-rate power source. Considerable amounts of power are needed to distribute, crush, elevate, and convey the raw material and the various products in any large reduction plant. Bearing friction involves waste, and serious efforts are being made in existing plants to avoid the extra expense; but the Hudson River company is probably the first of its kind in the field, handling rock or ore, in which every piece of moving equipment is provided with anti-friction bearings.

Neither the call of fashion nor an urge to create a record in plant-construction costs was responsible for the adoption of refinements such as those mentioned, and of many others described in the articles. Design throughout followed the dictates of good business; and the advantages that are certain to accrue include a satisfactory technical efficiency and high personnel morale, with low operating and maintenance cost—all of which advantages are reflected in benefits to consumers as well as to producers, in operating profits instead of operating losses. Of benefit to all concerned is the publication of information that stimulates additional enterprise of this character, that proves helpful in the correct application of a phase of engineering science and human experience to industrial progress.

Historical Summary

Engineering and Mining Journal was founded in 1866 as the *American Journal of Mining*, the title being changed in 1869. Publications with which *Engineering and Mining Journal* has been consolidated comprise: *Coal and Iron Record* (1875); *The Mining Review* (1877); *The Polytechnic Review* (1879); *Mining and Metallurgy* (1902); *The Mining Magazine* (1906); *Mining and Engineering World* (1917); *Mining and Scientific Press* (1922); *Engineering and Mining World* (1932).

The following have served as editors in chief of *Engineering and Mining Journal* and its predecessor publication: G. F. Dawson (1866-67); R. W. Raymond (1867-74); R. W. Raymond and R. P. Rothwell (1874-89); R. P. Rothwell (1890-1901); D. T. Day (1901-02); T. A. Rickard (1903-05); W. R. Ingalls (1905-19); J. E. Spurr (1919-27); A. W. Allen (1927-).

The editorial staff of *Engineering and Mining Journal* comprises: A. W. Allen, editor; A. H. Hubbell, H. D. Keiser, associate editors; W. N. P. Reed, assistant editor; H. H. Wanders, market editor.

Fairchild Aerial Surveys, Inc.

Large Crushing and Screening Plant Reflects Engineering Progress

Louis A. Slattery

Vice-President, Hudson River Stone Corporation

AN OUTSTANDING EXAMPLE of the latest in design and equipment—the large crushing, screening, and washing plant recently completed near Cold Spring, N. Y., by the Hudson River Stone Corporation, and costing $1,500,000—was started up on Nov. 28, 1932, after a two-day run without load to dry out the motors. The plant stands on the east side of the Hudson River, the upper or receiving end being close to the company's quarry, 300 ft. above the river, on the slope of Mount Taurus. The several buildings comprising it stretch down the mountain side in a line, 2,200 ft. long. They are connected by conveyor bridges, the lower or loading end extending across the narrow strip of level ground bordering the river, where trucks and railroad cars are loaded, and out over the shallows to deep water, to permit loading by barge. The side-hill site permits maximum advantage to be taken of gravity in handling the stone after the initial crushing. In all there are nine buildings, excluding power plant, office, and shops. They include No. 1, for primary crushing; No. 2, for secondary crushing; No. 3, a screen house, built over the storage "silos"; Nos. 4 and 6, wash houses; No. 7, truck-loading silo building; No. 9, car-loading silo building; and No. 8, barge-loading building.

Of the many features of the plant, the following are of especial interest:

1. With its capacity of 1,000 tons per hour of finished product, 2¾ in. and smaller, it is the largest crushing plant producing concrete aggregate and ballast rock.

2. A diesel-driven power plant, capable of developing 1,980 hp., makes it independent of purchased power. It is probably the only crushing plant, producing stone for commercial purposes and of more than 2,500 tons' daily capacity, to have its own auxiliary power plant, which in turn is the largest to be installed to date in conjunction with a crushing plant of this nature.

3. It is the only rock-crushing plant that washes the product before and after storage.

4. Considerable economy in power consumption is effected because every piece of moving equipment is provided with anti-friction bearings.

5. An apron feeder is used to deliver raw material to the primary crusher.

6. Means are provided for returning material larger than 1¼ in. from the screen house directly to the crushing plant, in the absence of immediate demand, without the operators' being compelled to store the rock first.

7. Every conveyor has a V-belt driven speed reducer mounted on one massive base plate that ties head shaft and countershaft.

8. Construction is fireproof throughout. No wood has been

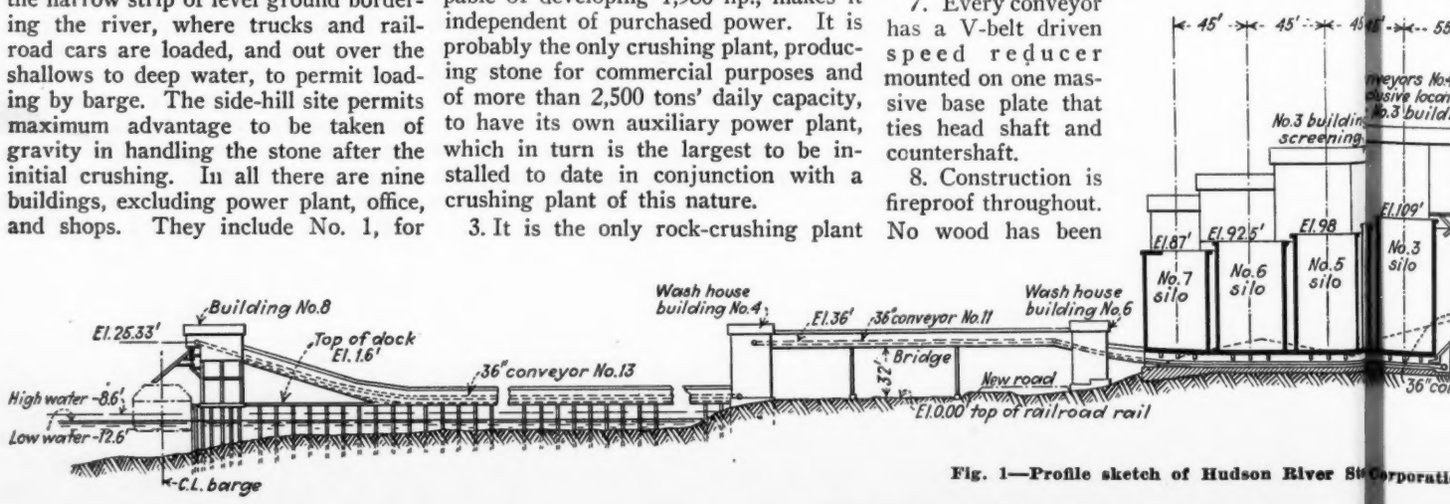


Fig. 1—Profile sketch of Hudson River St Corporati

used. Building materials employed have been limited to concrete, structural steel, and copper-bearing galvanized-steel siding and roofing.

The stone crushed is Storm King granite, close-grained, hard, and rather abrasive. Specifically, its characteristics are: crushing strength, 20,000 to 32,000 lb. per sq.in.; French coefficient of hardness, 11.5 to 13, with a percentage of wear or abrasion by this test of 2.25 to 3.25; water absorption, under 0.5 per cent. These meet the specifications of the various highway and other departments of the state, county, and municipal governments of the area the plant will naturally serve, including those of New York City and the Port of New York Authority.

Products of the plant are crushed and washed stone in several sizes, each of which is stored in a separate silo or concrete bin prior to shipment. These sizes and the respective silos, the latter designated by the numbers used at the plant and in this article, are:

— 2½	in. +	2½	in.	No. 1 silo
— 2½	in. +	1½	in.	No. 2 silo
— 1½	in. +	1¼	in.	No. 3 silo
— 1¼	in. +	1	in.	No. 4 silo
— 1	in. +	⅝	in.	No. 5 silo
— ⅝	in. +	⅜	in.	No. 7 silo
— ⅜	in. +	¼	in.	No. 6 silo

Minus-¼-in. material is discarded, being discharged into a settling cove along the river bank.

Primary Crushing

Fig. 2 shows the arrangement of the primary crushing equipment and accessories in Building No. 1. The equipment includes:

A heavily built open feed hopper
A 72-in. (wide) by 15-ft. (long) Traylor extra-heavy steel-link apron feeder
One 56x72-in. Traylor Bulldog cast-steel jaw crusher
A chute box which delivers jaw-crusher discharge to following gyratories
Two 30-25-in. Traylor BH Bulldog gyratory crushers
One 25-ton hand-propelled geared Traylor trolley hoist
One 25-ton Shepard-Niles hand-propelled traveling crane
Tail end of conveyor No. 1

Conveyor Data—Hudson River Stone Corporation's Plant, Cold Spring, N. Y.

No.	Width In.	Ply	Length Ft. c. to c.	Motor Hp.	Size of Rock	Capacity, Tons/Hr.	Grade
1.....	42	7	240	20	4 in. to dust	1,000	Level
2.....	42	7	292	15	2½ in. to dust	1,050	—2 deg.
3.....	24	6	315	40	1½ in. to 2½ in.	300	+ 17 deg.
4.....	36	7	100	20	2½ in. to dust	800	+ 5½ deg.
5.....	30	6	69	7½	1½ in. to 2½ in.	300	+ 3 deg.
6.....	30	6	99	15	1½ in. to dust	500	+ 14 deg.
7.....	24	6	58	5	1 in. to 1½ in.	150	Level
8.....	24	6	53	15	⅜ in. to dust	300	+ 13 deg.
9.....	24	6	57	5	⅜ in. to ¾ in.	100	Level
10.....	36	7	50	7½	2½ in. to 2½ in.	1,000	— 8 deg.
11.....	36	7	470	40	2½ in. to dust	300	Part Level Part + 12 deg.
12.....	24	6	270	15	2½ in. to dust	1,000	Part level Part + 12 deg.
13.....	36	8	670	75	2½ in. to dust	1,000	Part level Part + 15 deg.
14.....	24	6	178	30	⅜ in. to 2½ in.	300	+ 17 deg.
15.....	36	7	75	40	¾ in. to 2½ in.	1,000	+ 17 deg.

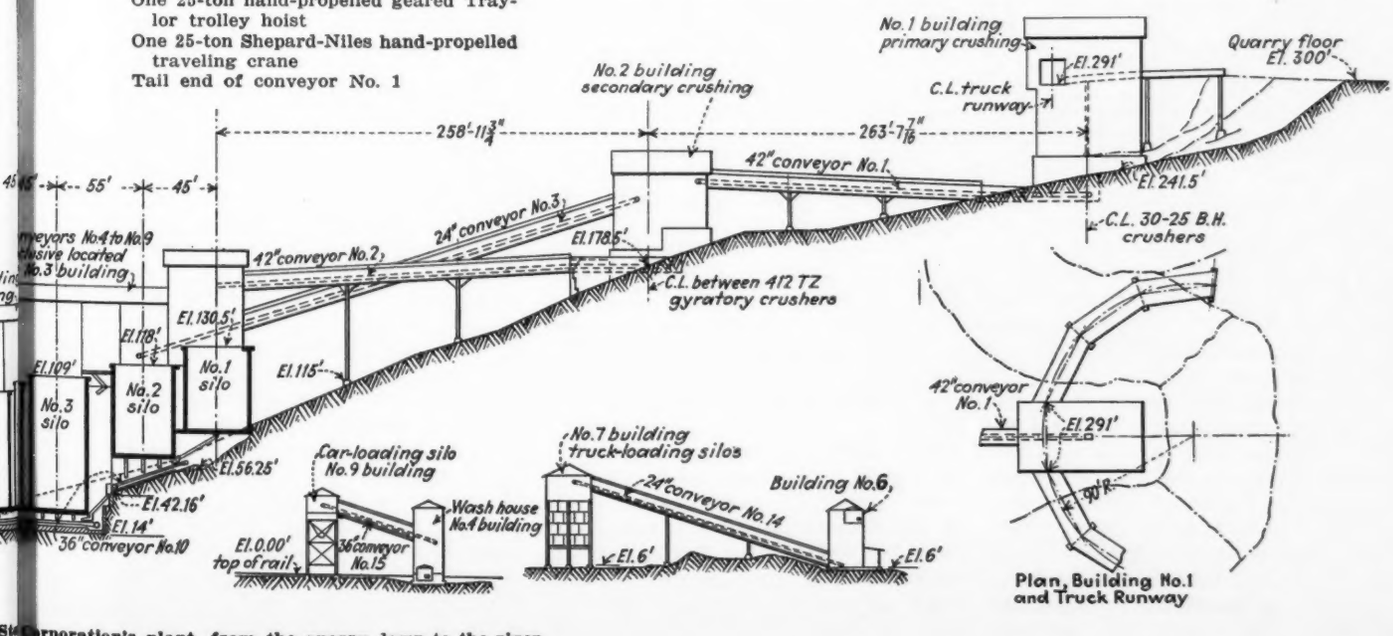
Stone dumped by trucks into the hopper above the apron feeder is fed uniformly to the jaw crusher. Thanks to the feeder practically no bridging of large pieces occurs. The crusher is set to 12 in. It discharges into a heavily built lined box, which is supported on the same foundations as the jaw crusher and delivers the stone, together with the chips and fines that fall between the feeder links, to the two gyratories mentioned. Two doors provided in this box make it possible to cut out either of the gyratories when the plant is running at half capacity or when a gyratory must be shut down for any other reason. Each gyratory can handle the jaw crusher's product at a rate of 500 tons per hour when set to 3½ in. The gyratory discharge, 3½ in. in size and smaller, falls on a 42-in. horizontal troughing belt conveyor, known as No. 1, which delivers it for further crushing to Building No. 2.

Further details regarding the equipment in Building No. 1 are warranted. The feeder is mounted on a heavy steel frame and is driven at a speed of 30 ft. per minute by a 30-hp., 1,200-r.p.m. induction motor through a direct-connected speed reducer and a roller bushed chain. This chain will make it possible to speed up the feeder to increase the plant capacity, if this should ever be desired.

The jaw crusher is built in sections, with sides and ends in separate pieces. Despite this, some of the castings approach 40 tons in weight. Moving these heavy parts from the railroad to the point of installation high up the hillside required special means. Shipped from the factory in special deep-well cars, they were lifted out one at a time at Cold Spring by the railroad's wrecking crane and set on low many-wheeled motored trailers. Thus the long upgrade haul was made, with a 30-ton tractor in front supplementing the motor, and another behind, pushing. Curves in the road were negotiated with the aid of snatch blocks.

Design of the crusher is modern, including water-cooled bearings and manganese-steel crushing and cheek plates. It is driven at 100 r.p.m. by a 250-hp., 600-r.p.m., slip-ring motor and a 36-in. transmission belt that is made of extra heavy rubber.

The two gyratories following the jaw crusher have heads and concaves, as well as diaphragm liners and hopper



Stone Corporation's plant, from the quarry down to the river

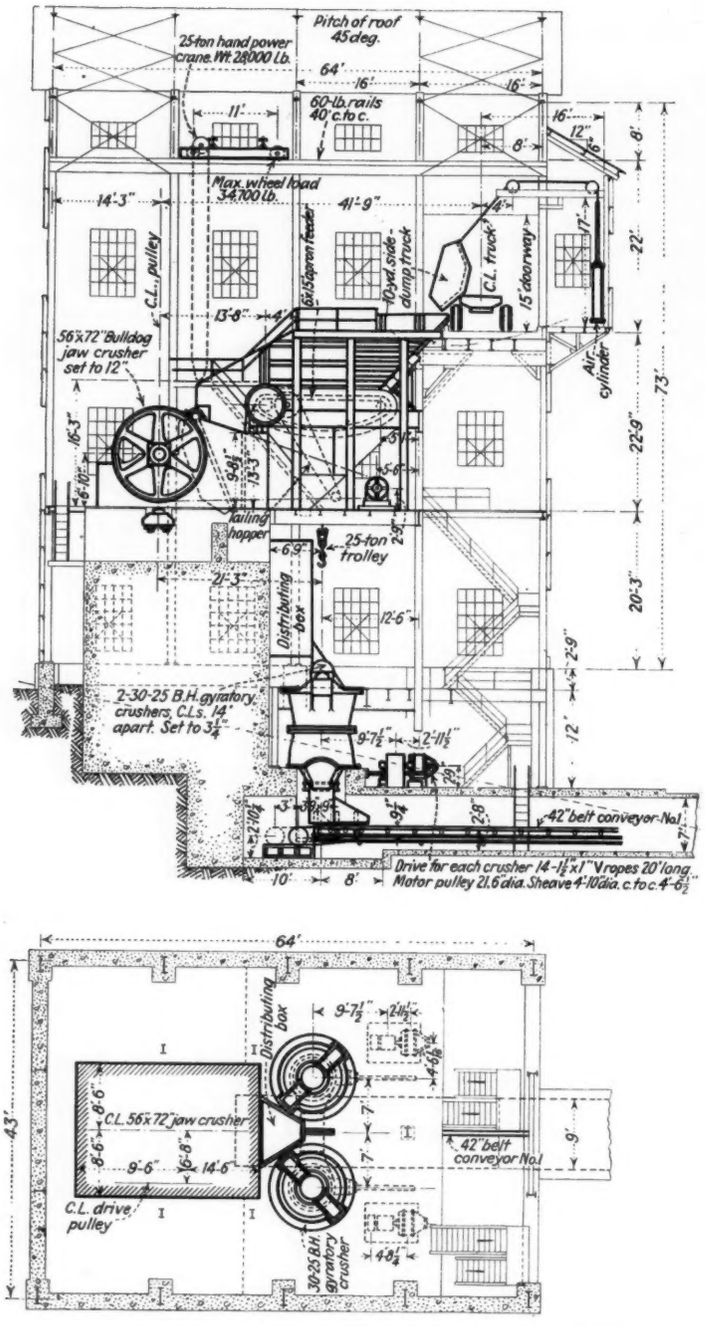


Fig. 2—Plan and elevation of Building No. 1, primary crushing, Hudson River Stone Corporation

liners, of manganese steel. The heads are bell-shaped and the concaves inverted. Shell and spider are of cast steel. Each crusher is driven by a 200-hp., 900-r.p.m. three-bearing slip-ring motor through a heavy short-center V-rope drive, which gives the head about 125 gyrations per minute.

For servicing jaw crusher and feeder the 25-ton crane is used. Should the stone bridge above either, an air-cylinder hoist can be hung on the hook with tongs for loosening the mass. Gyratories and motors are served by the 25-ton trolley hoist. The I-beam on which this travels runs out through the side wall, enabling it to move equipment into or out of the building. The crane can similarly handle parts loaded on trucks on the elevated runway.

No. 1 conveyor, in transporting the gyratory product to Building No. 2, passes first through a 60-ft. tunnel and then through a gallery which enters the top of the next building, as the illustrations show. Only in width and length does it differ from any of the other conveyors.

The plant has fifteen conveyors. Their total length, each being measured center to center, is approximately 3,100 ft. Total length of conveyor belt in the plant is 6,400 ft. The belting was all made to special specifications, defining width, ply, weight of duck, thickness of rubber covering, and kind of rubber and of cotton, and covering tensile density, friction, and cover pull.

A unique feature of the plant is the type of conveyor drive employed. This is the same for each conveyor, though the drive dimensions may differ. In each case, the head end of the conveyor is mounted on a Traylor inclosed speed-reducer unit designed so that the head shaft and the countershaft are on one base plate. The conveyor is driven by an induction motor through the medium of a multiple V-belt drive between the pulleys of the motor and speed reducer. The data for the fifteen conveyors and their drives are given in the table on the preceding page.

The purpose in installing this drive is to make it possible to increase or decrease a conveyor's speed, if this becomes desirable, simply by changing the V-rope drive and leaving motor and speed reducer unchanged. Conveyor speed now ranges between 250 and 400 ft. per minute, the larger capacity belts running at the higher speed and some of the smaller belts between units, carrying small tonnage, at slower speeds. This arrangement made it possible to use belts of only four widths.

To prevent conveyors set on an incline from running away, should motor or power fail, each drive is fitted with a backstop.

Secondary Crushing

Building No. 2 has the following major equipment:

- Two heavily built 72-in. x 16-ft. open-end Traylor revolving scalping screens and dust jackets
- Two baffle-screen wash boxes, with water line
- Three Traylor No. 412 TZ Bulldog finishing gyratory crushers. Space has been allowed for an additional unit.
- Conveyor No. 1 delivering; No. 2 receiving

Inasmuch as the crushed stone delivered to Building No. 2 from No. 1 has the overburden from the quarry mixed with it, a thorough washing is necessary before further handling. The conveyor discharge is divided by a two-way spout between the two scalping screens, which are fitted on the inside either with screen plates having 2¾-in. round holes, if ballast stone is to be made, or with plates having smaller perforations, if ballast is not wanted. The dust jacket on the outside of each screen has ⅞-in. round holes. Wash water is applied under 50-lb. nozzle pressure in the chutes leading to the screens and inside the screens by means of perforated spray pipe. The water, dirt, and other fine material passing through the ⅞-in. holes of each dust jacket go into a steel wash box underneath, which has three wire-mesh baffle screens with ¼-in. square openings. Wash water under 50-lb. nozzle pressure is applied at these baffles also. The water and dirt discharged from the wash boxes run either by a launder directly to the

settling cove or to the sump of an 8-in. sand pump in wash house No. 6, which pumps it to the cove. The —$\frac{7}{8}$-in. +$\frac{1}{4}$-in. washed stone from the boxes falls into a box feeding No. 2 conveyor in the bottom of the building, which also receives the —$2\frac{3}{4}$-in. +$\frac{7}{8}$-in. stone from the scalping screens. The +$2\frac{3}{4}$-in. oversize from the latter falls into the distributing box feeding the three finishing gyratories, which are set to crush within a range of $\frac{3}{4}$ in. to $1\frac{1}{2}$ in. Their product also goes on to No. 2 conveyor, with the other sizes of stone mentioned, so that the material carried by this conveyor, which goes to the screen house, or Building No. 3, ranges from +$\frac{1}{4}$ to —$2\frac{3}{4}$ in. in size.

Each scalping screen is driven at about 12 r.p.m. by a 25-hp., 600-r.p.m. slip-ring motor through a heavy V-belt short-center drive.

Foreseeing a time when demand for stone larger than $1\frac{1}{2}$ in. may cease, the company has provided room for one more finishing gyratory in Building No. 2. Those already installed are oil-cooled and, like the two gyratories in Building No. 1, have head, concaves, and diaphragm and hopper liner plates of manganese steel and shell and spider of cast steel. Each is driven by a 125-hp. 900-r.p.m. slip-ring motor through a short-center, V-belt drive which gives the head about 200 gyrations per minute. Any of them can be shut down separately, after closing the corresponding openings in the distributing box.

To service the revolving screens a 2-ton chain block is provided; likewise a 10-ton block for the gyratories. Each is so mounted that it can load or unload trucks carrying equipment parts.

No. 2 conveyor runs to the top of Building No. 3 through an inclined trestle-supported gallery. For about 65 ft. its tail end is housed in a tunnel under Building No. 2. Details as to drive are given in the table.

Screening

Building No. 3, or the screen house, is approximately 325 ft. long and is built on top of the seven storage silos, each of which is of concrete and approximately 40 ft. in diameter. In addition to the discharge end of No. 2 conveyor, its principal equipment is as follows:

Seven troughing belt conveyors, Nos. 3 to 9 inclusive, and their respective drives
Fourteen Niagara 4x9-ft. double-deck vibrating screens and their drives
Two Niagara 4x9-ft. single-deck vibrating screens and drives
Distributors and chutes

The arrangement of the seven conveyors and the screens, together with the discharge end of No. 2 conveyor, is shown in the diagrammatic sketch in Fig. 4. A more detailed drawing, showing elevations and vertical sections of the building over Nos. 2 and 3 silos, is presented in Fig. 5. The arrangement of the screens over silo No. 3 is sketched in Fig. 6. These details are typical of the screen house in general.

A diagrammatic flowsheet of the screening and storing operations is also shown in Fig. 4. From this and the sketch above it one can see that screening is done only over Nos. 1, 3, 5, and 6 silos. The other silos receive their respective stone sizes by conveyor from the screening points. No. 3 conveyor is unique in that its function is to take back to the secondary crushing plant for recrushing certain sizes of stone, whenever the silo, or silos, in which they would be stored (Nos. 1, 2, and 3) are full, or whenever no more stone of these sizes is wanted for the time being, owing to lack of demand. The means of bypassing such stone to this conveyor is shown in Fig. 5.

Vibration of the screen house structure has been reduced to a minimum by suspending all of the sixteen screens by cable and shackle from overhead supporting steel members, so that each with its motor is free to swing. Each screen is driven by a 5-hp. 1,800-r.p.m. induction motor through a V-belt, the motor being fastened to angle-iron support attached to the screen frame. A 2-ton chain block and trolley are provided in this building for lifting equipment parts.

Thirty thousand tons' storage capacity is afforded by the seven concrete silos. Each of the seven is of 40-ft. diameter and has 9-in. walls reinforced horizontally and vertically. Their height depends on the contour of the ground, varying from 55 to 90 ft. above their foundations (Fig. 1). Nos. 1, 2, and 3 are built on solid rock, the others

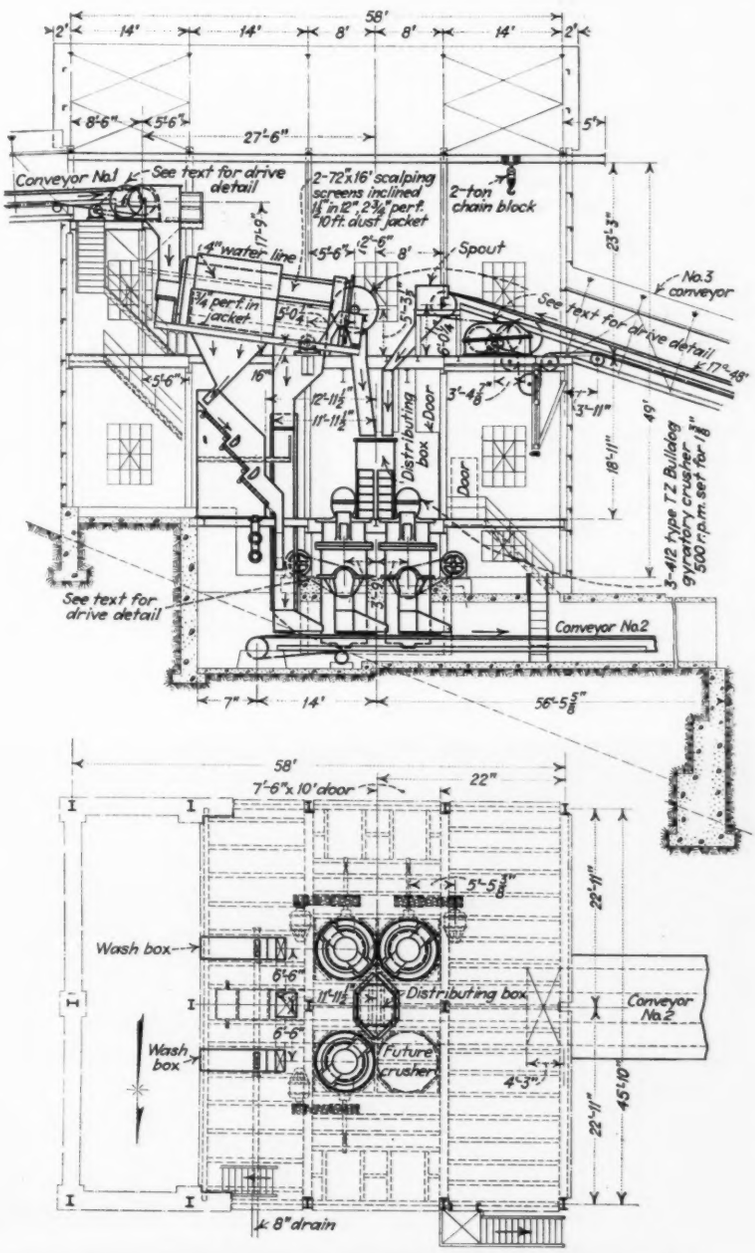
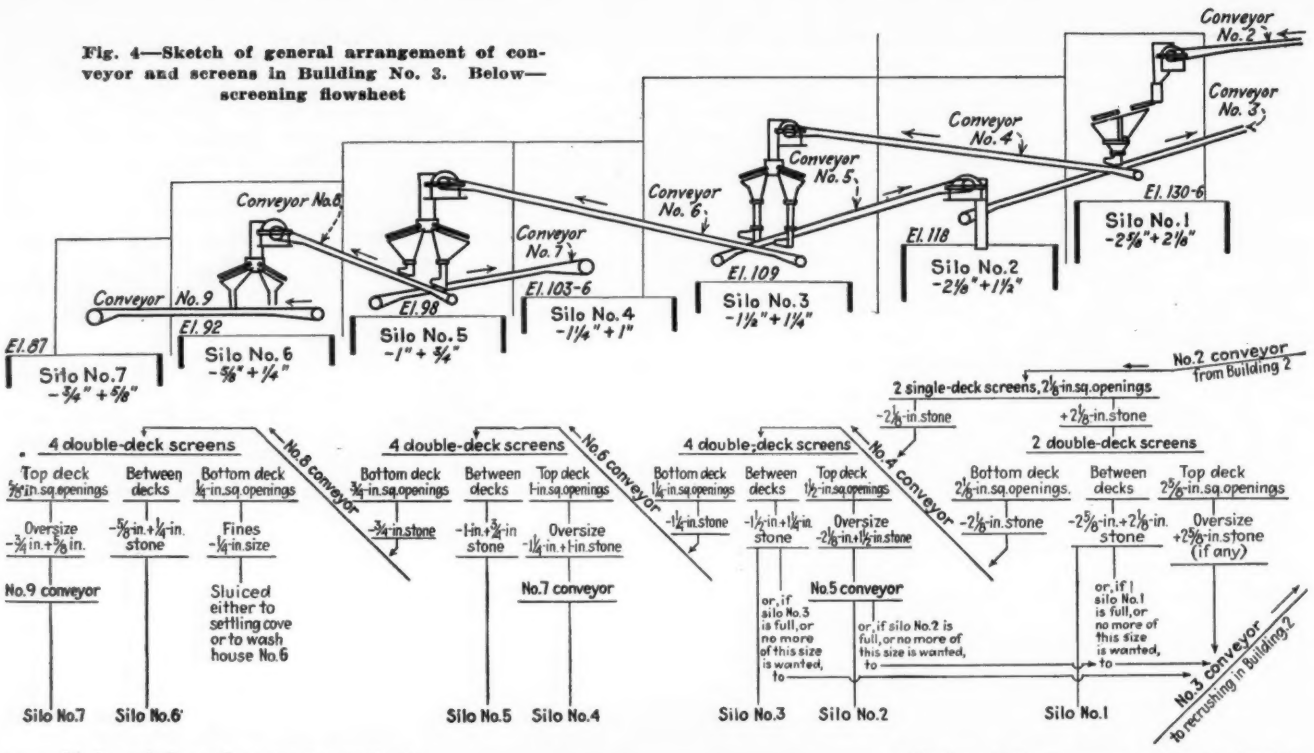


Fig. 3—Building No. 2, secondary crushing. Plan and elevation

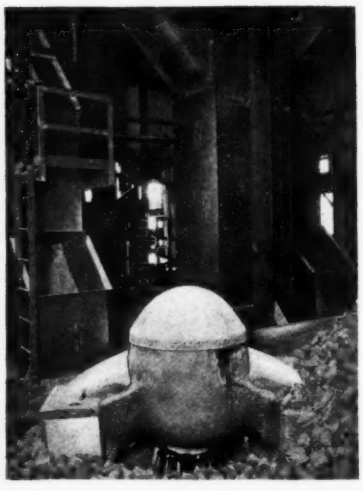
Fig. 4—Sketch of general arrangement of conveyor and screens in Building No. 3. Below—screening flowsheet



on a 3½-ft. reinforced concrete mattress foundation which ties the four together and rests on good soil.

Loading

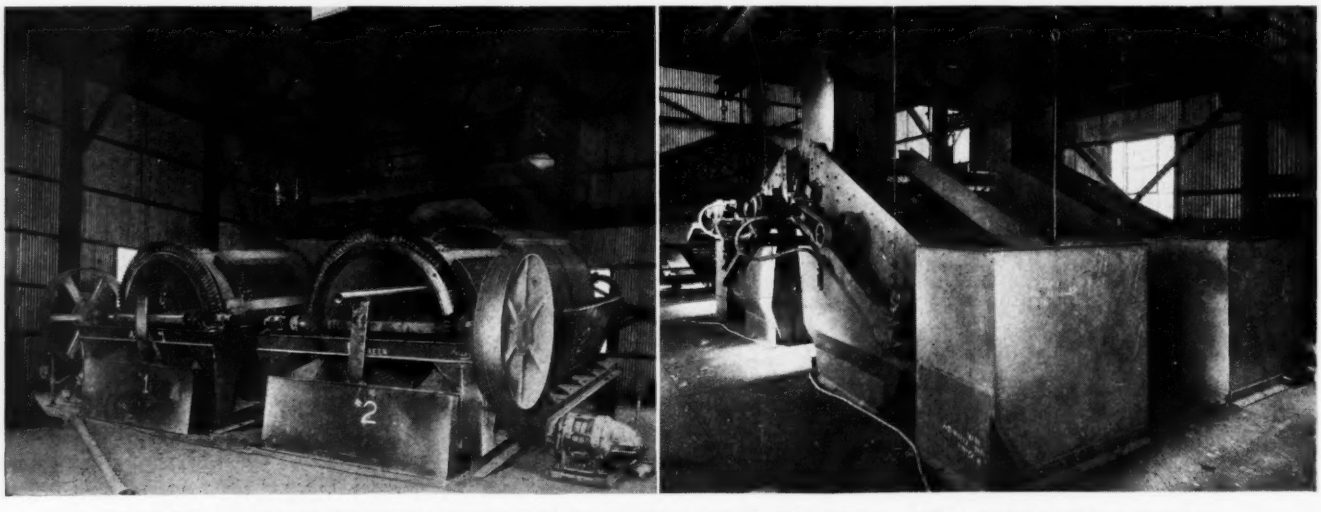
To load trucks, cars, or barges, the various stone sizes are drawn from their respective silos through bottom gates which, except under Nos. 1 and 2 silos, deliver to two belt conveyors, Nos. 11 and 12. These conveyors run in a 7x11-ft. tunnel built on top of the foundations of silos Nos. 3 to 7 inclusive, as shown in Fig. 7. No. 1 silo has only one bottom gate. Stone drawn from it is passed by a long chute of large opening to a conveyor, No. 10, which runs under No. 2 silo. At present this silo has only three gates, which deliver onto the conveyor. Eventually, if demand for the size of stone to be stored in this silo warrants, another conveyor similar to No. 10 will be installed and three more gates to feed it. No. 10 conveyor delivers to a box which feeds either No. 11 or No. 12 conveyor or both of them.



One of the three secondary gyratory crushers in Building No. 2. Wash boxes in rear.

The walls of the tunnel under Nos. 3 to 7 silos are of reinforced concrete 15 in. thick, the roof of concrete 24 to 30 in. thick. These five silos have six gates each, three over each conveyor. The gate boxes, 24 in. square, are built in the concrete tunnel roof and are fitted with swinging gates specially designed to permit either full or partial flow from each. By such means, several sizes of stone can be mixed on one belt.

When trucks are to be loaded, the crushed stone is drawn off from the silos in the sizes desired, on to No. 12 conveyor, which delivers it to No. 6 wash house, shown in Fig. 8. Here it is allowed to fall through a steel wash box having ¼-in. screen baffles, adjustable as to inclination, such as are used



Left, Revolving scalping screens that precede the gyratories in Building No. 2. Right, Battery of four vibrating screens in screen house, Building No. 3.

in Building No. 2, already described. Spray pipes with water at 50-lb. nozzle pressure give it a thorough final cleaning. It is then carried by No. 14 conveyor to the truck-loading silo building, No. 7, and dumped into a swivel chute which directs it into any one of four 100-ton steel bins, or silos, 12 ft. in diameter and 18 ft. high. These are set high enough above the ground to permit trucks to be loaded beneath them through bottom duplex gates and side swing gates. Eight trucks can be loaded at once.

The fines removed in this washing, together with the dirty water, flow into a steel box from which they are pumped to the settling cove by an 8-in. centrifugal sand-dredge pump, which is direct-connected to a 75-hp., 515-r.p.m. slip-ring motor. If needed, another similar pumping unit can be installed.

For loading railroad cars or barges, the stone is drawn from the silos on to No. 11 conveyor, which parallels No. 12 conveyor as far as No. 6 wash house and then, continuing on, crosses the

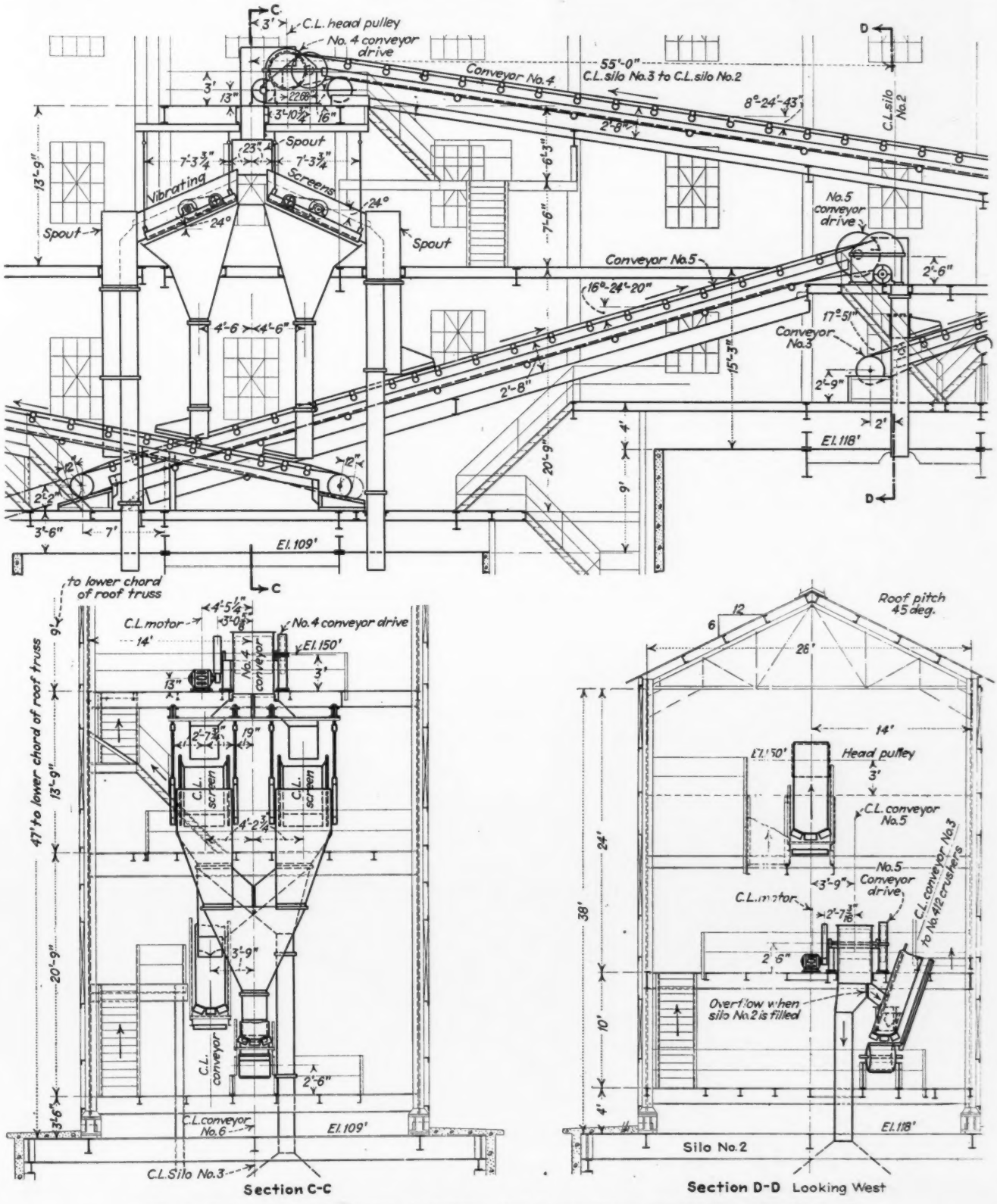
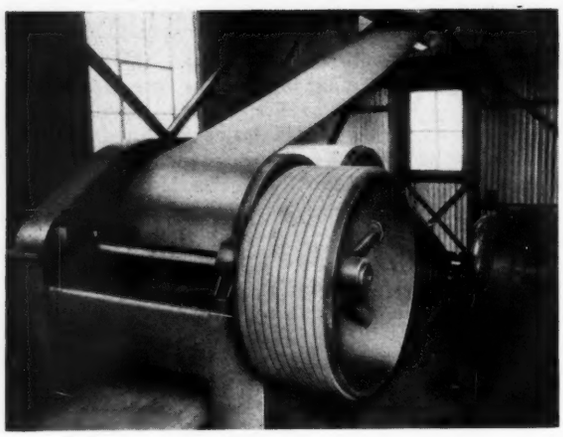


Fig. 5—Part of the screen house, or Building No. 3, over Nos. 2 and 3 silos. Elevation and vertical cross sections. Hudson River Stone Corporation



Typical conveyor drive in Hudson River Stone Corporation's plant, combining a multiple V-belt drive with a speed reducer which ties head shaft and countershaft on a common base plate. No. 13 conveyor is shown

highway and the railroad to No. 4 wash house. Here there are two steel wash boxes like that in No. 6 wash house, also with adjustable screens. The dirty water and fines flow to the near-by settling cove. After being washed, the stone is conveyed either by No. 15 conveyor to the railroad ballast bin (Fig. 10) or by No. 13 conveyor, known as the dock belt conveyor, which runs in a steel gallery over a wood pile trestle, to a swinging chute in Building No. 8 loading barges (Fig. 11).

Water used in the various washing operations is supplied by two 8-in. centrifugal pumps housed in Building No. 8 (Fig. 11). One, a single-stage unit, capable of handling 2,000 g.p.m. at 100-lb. pressure, furnishes the water for Nos. 4 and 6 wash houses. It is driven by a direct-connected 150-hp., 1,800-r.p.m. synchronous motor, with a direct-connected exciter. When loading only is under way, this pump is used. The other is two-stage, of 1,000-g.p.m. capacity at 200-lb. pressure, and supplies wash water for the scalping screens in Building No. 2 and the last set of vibrating screens in Building No. 3. When crushing and screening only are being done, and no loading, it is in service. It is driven by a 150-hp., 1,800-r.p.m. synchronous motor with direct-connected exciter. These are the only synchronous motors in the plant. Both pumps are required when crushing, screening, and loading are all under way.

pany, Inc. Construction, including all concrete work, excepting that of the silos and the steel work, was done by the Hudson River Stone Corporation under the direction of Louis A. Slattery and Walter Caccia, chief engineer, Lyons-Slattery Company, and under the direct supervision of R. A. Gainey, general superintendent; A. Craig, assistant engineer; and O. J. de Gaetano, electrical engineer. Equipment was handled and installed by the Stone Corporation, the Traylor company supervising.

The plant equipment was supplied by the following manufacturers:

Crushers, apron-feeder, revolving screens, speed reducer units, bin gates, bin boxes, chutes, hoppers and liners: Traylor Engineering & Manufacturing Company

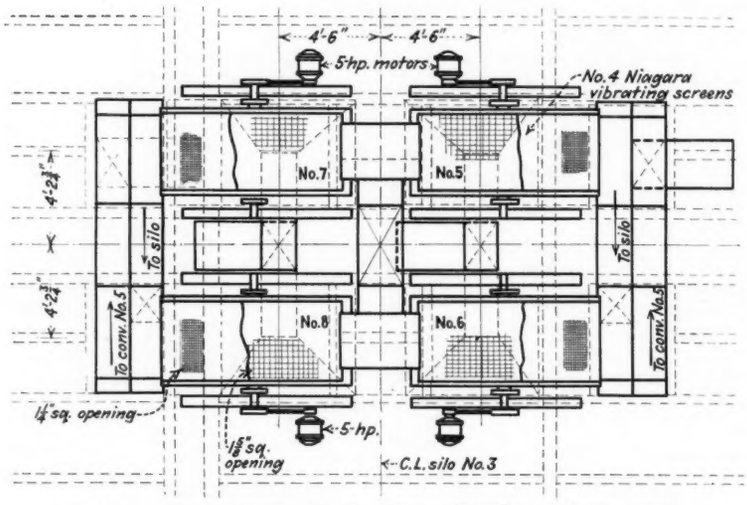


Fig. 6 — Arrangement of the four vibrating screens over No. 3 storage silo

With exception of the diesel power plant, the general layout of the crushing plant was designed by the Traylor Engineering & Manufacturing Company, in conjunction with Lyons-Slattery Com-

Generators, motors, control equipment, transformers, switchboards and panels: General Electric Company

Belt-conveyor equipment, such as troughing and return rollers, head and tail iron equipment: Chain Belt Company

Vibrating screens: Niagara Concrete Mixer Company

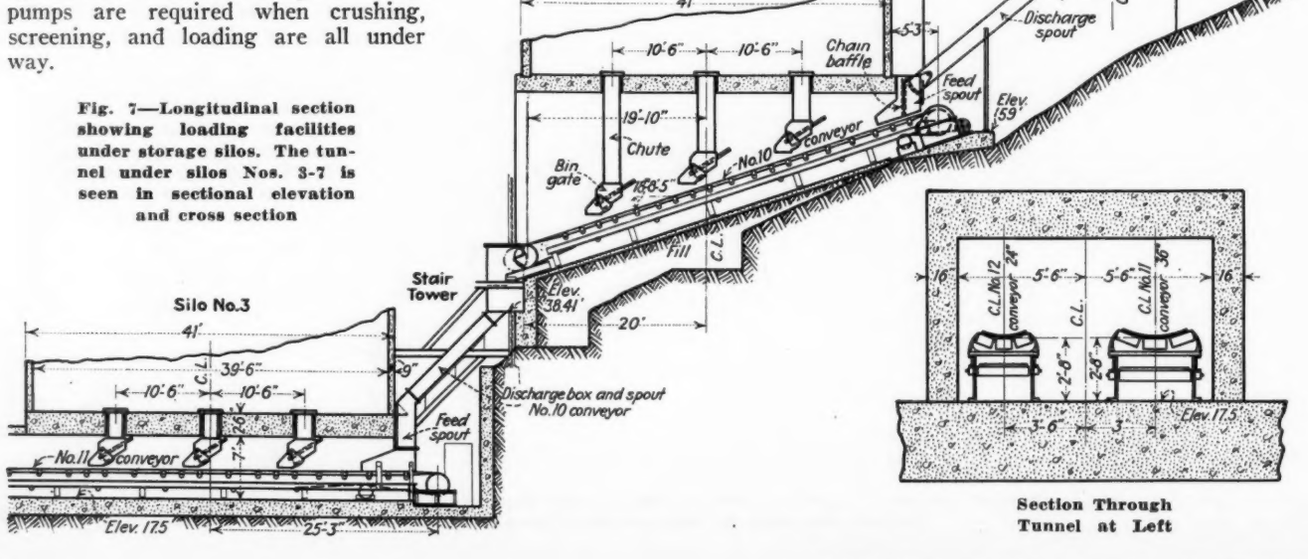


Fig. 7—Longitudinal section showing loading facilities under storage silos. The tunnel under silos Nos. 3-7 is seen in sectional elevation and cross section

Section Through Tunnel at Left

Anti-friction bearings, dust-proof: principally Timken Roller Bearing Company; also S.K.F. Industries, Inc., and Dodge Manufacturing Company
Conveyor belting: Quaker City Rubber Company and New York Rubber Company
V-rope drive belts: Gates Rubber Company
Traveling crane, 25-ton, in Building No. 1 and 3-ton crane in power house: Shepard-Niles Crane & Hoist Company
Centrifugal water pumps: Goulds Manufacturing Company
Centrifugal sand-dredge pump: Morris Machine Works
Diesel engines and accessories: McIntosh-Seymour Corporation
Air cylinder hoists: Curtis Pneumatic Machinery Company
Chain blocks and trolleys: Reading Chain & Block Corporation
Steel buildings, truck storage tanks, corrugated roofing, and siding: American Bridge Company (designed and erected).
Oil storage tanks: Buffalo Tank Company
Piping, valves, fittings: Pittsburgh Piping & Equipment Company
Armored cable: General Cable Company
Concrete silos: Macdonald-Spencer Engineering Company

The Quarry

The company's quarry, at present of small dimensions, is on the mountain side, 300 ft. above the railroad track and close to the primary crushing plant. It is being opened by side-hill cuts, so that the height of face will gradually increase. The floor of the quarry is level and only 9 ft. higher than the jaw-crusher dumping floor. An enormous tonnage of granite is in sight. Overburden varies in depth and for the present is being shot down with the granite and sent with it into the plant.

The stone is loaded by steam shovels into Mack trucks having 10-yd. side-dumping Easton bodies. Three such trucks are in use. As operations warrant the number will be increased. They enter the crushing plant by the curved runway shown and are dumped by means of a hook and air cylinder. A new Marion shovel, Model 125, full-revolving, caterpillar-mounted, and equipped with a 3½-yd. manganese-steel dipper and a 38-ft. boom, has only recently been assembled. It can be seen in one of the illustrations, as well as a smaller, full-revolving, caterpillar-mounted Osgood shovel, having a 1½-yd. dipper. A caterpillar-mounted crane completes the quarry equipment of this class.

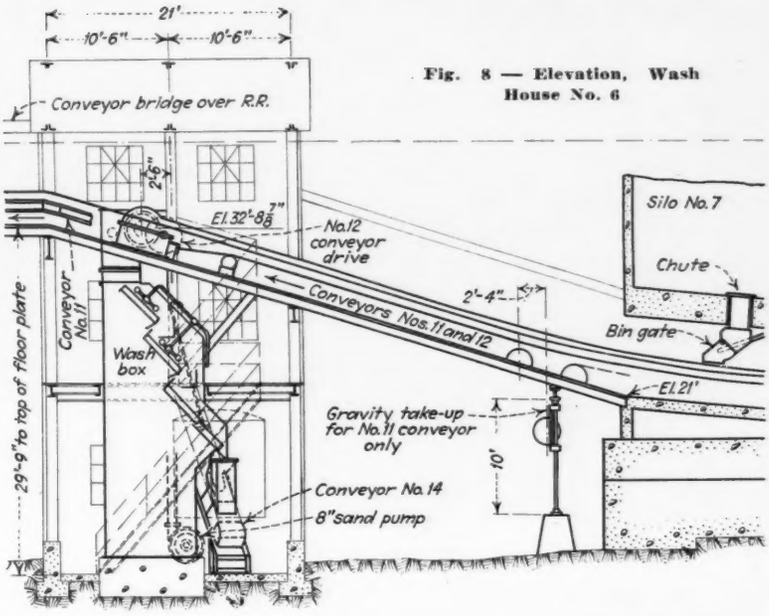
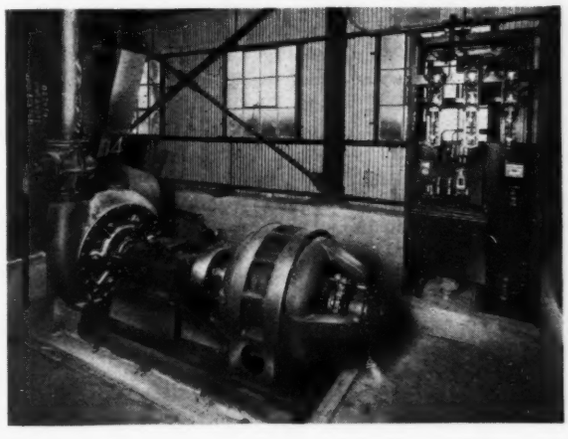


Fig. 8 — Elevation, Wash House No. 6

Drilling for blasting purposes is in an experimental stage. Later the practice can be more definitely discussed. For the present, it is being done in three ways: by well-drill holes; by drilling holes, more or less flat, into the quarry face; and by driving coyote tunnels and drifts. Well holes, 8-in. in diameter, of a depth depending on the height of bank, and of a spacing and burden determined by the condition of the face, are put down with a Cyclone rig. These holes are shot with cordeau in the standard way, electrically fired with a single cap. The second method, involving drilling of the quarry face, is done with No. 71 water Leyner drifters, tripod-mounted and drilling 22-ft. holes 2½ in. in diameter—one row collared about 5 ft. apart just below the quarry grade and pitched down somewhat, and, 4 or 5 ft. above, a second row of 22-ft. holes pitching upward, the pitch depending on the ground above. These holes are shot with 60-per cent gelatin and electric blasting caps, being sprung once or twice if the ground will stand it. This has proved sufficient to break the moderate heights of face thus far encountered, for the full depth of the round.

Coyote tunnels are driven 3x4 ft., untimbered, with IR-74 wet drifters mounted on column and arm and using 1-in. hollow hexagon steel. A round of eleven 2.5 to 3-ft. holes is used, electrically fired, with delays on all except three center holes. Three men compose a crew, one mucking and the



Eight-inch sand pump in wash house No. 6. Control board at right. Conveyor No. 14 in background

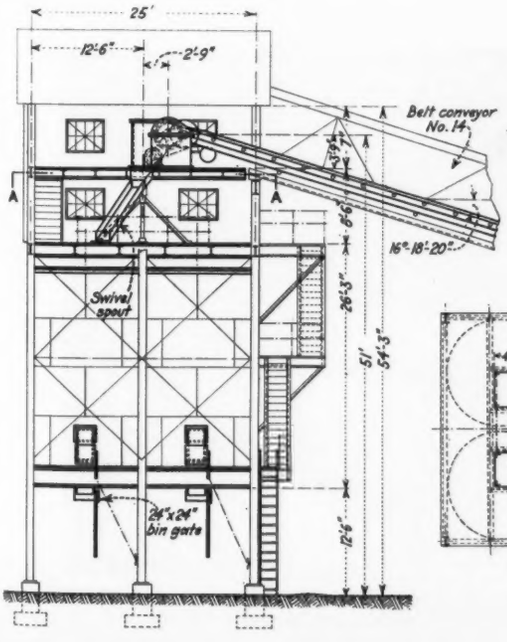
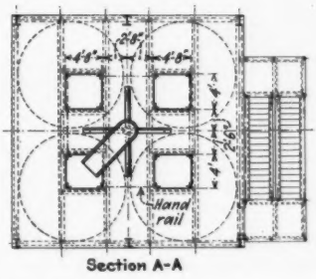


Fig. 9 — Truck-loading silo. Elevation and section



Section A-A

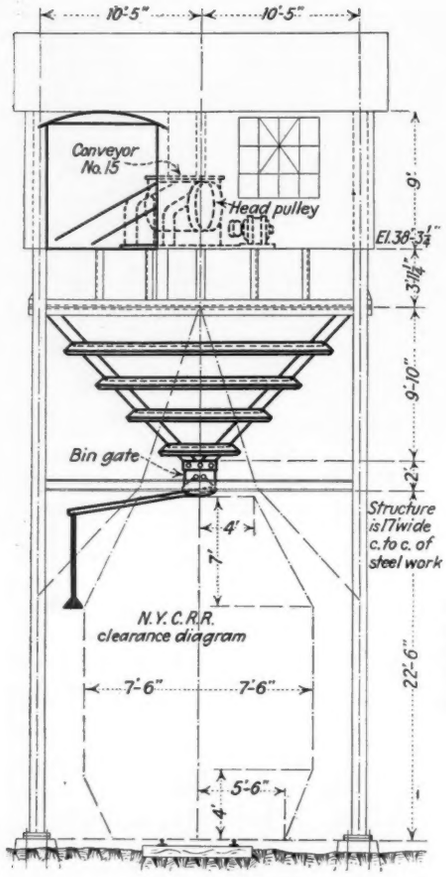


Fig. 10—Car-loading or No. 9 building. Elevation

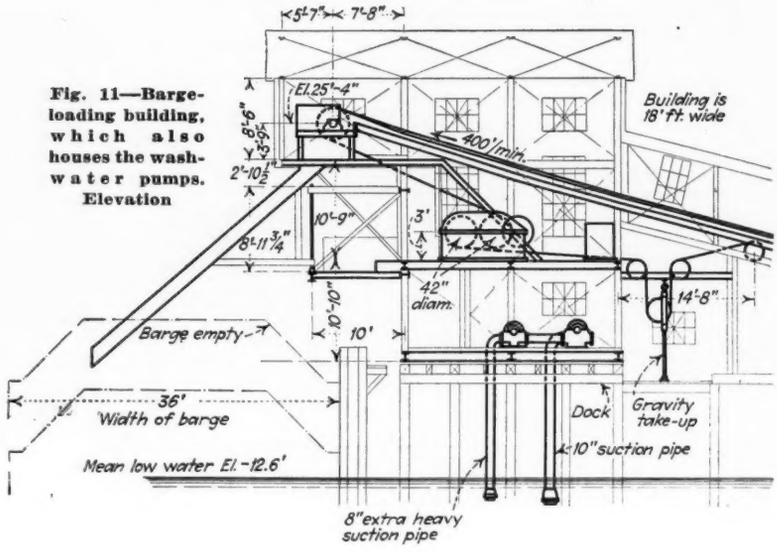


Fig. 11—Barge-loading building, which also houses the wash-water pumps. Elevation

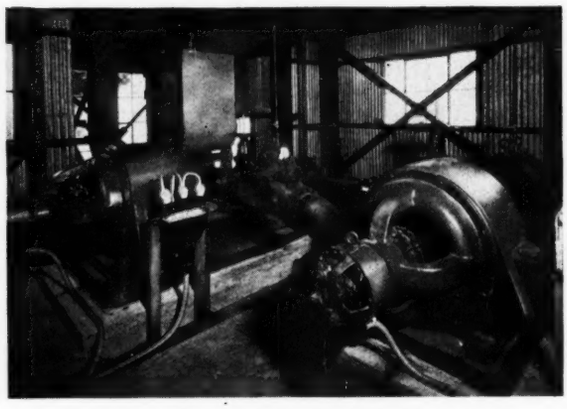
others drilling. One blast, thus prepared, has been fired so far. It involved 155 ft. of tunnel and broke about 45,000 yd. with 6¼ tons of 60 per cent gelatin. Preparations for a second blast involving 570 ft. of tunneling (including a 90-ft. stem, two 115-ft. wings, and two 125-ft. wings) have been completed. It is expected to break about 200,000 yd. (2¼ tons/cu.yd.). Loading is done in the usual manner, cordeau being used, supplemented by a separate electric blasting circuit.

In driving the tunnels and in drilling the 22-ft. holes in the quarry face, Thurston detachable bits, in ten gages, from 2¾ in. down, are being used and are giving satisfaction.

To date, excluding the last blast mentioned, about 65,000 cu.yd. of rock has been broken. Enough will be put through the plant on the present run to meet certain orders and fill the storage silos.

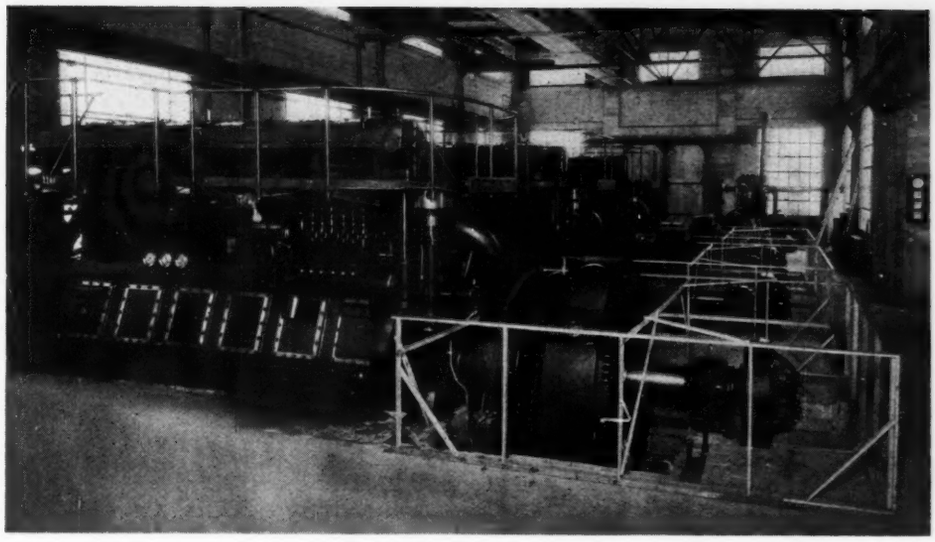
A rip-rap quarry has also been opened on a point of land on the river's edge, close to the power plant. Here 8-in. well holes, 60 ft. deep, are shot with 40 and 60 per cent gelatin (DuPont Gelex A and No. 1), cordeau and electric firing being used. The method of loading employed differs from that used in the quarry on the hill because large fragments are desired.

The Hudson River Stone Corporation was organized in the early part of 1931, and is owned jointly by O'Brien Brothers Sand & Gravel Company and Lyons-Slattery Company, Inc., contractors, both with headquarters in New York City. J. J. O'Brien is president; Louis A. Slattery, vice-president; M. J. Lyons, treasurer; J. V. O'Brien, assistant treasurer; T. F. O'Brien, Jr., secretary.



Two wash-water pumps and their motors in barge-loading building on dock

One of the trucks with stone from the quarry ready to dump into hopper of jaw crusher apron feeder



Power house, Hudson River Stone Corporation, Cold Spring, N. Y. It houses three 500-kw. generators, each driven by a 660-hp. four-cycle solid-injection diesel engine. A fourth unit will ultimately be installed

Low Power Costs From Diesel-Electric Plant

O. J. de Gaetano

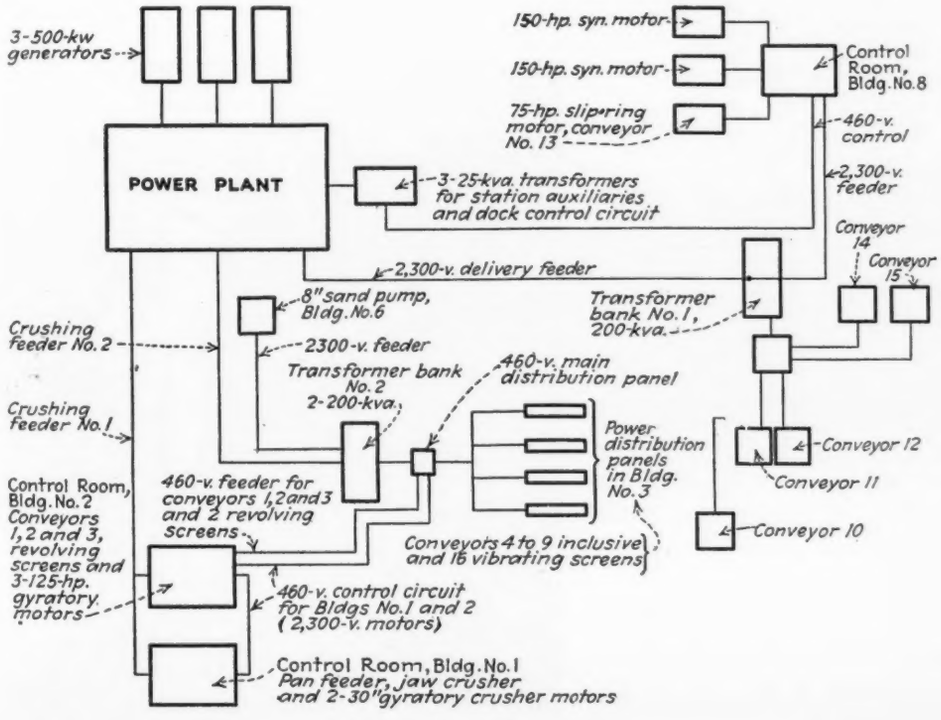
Electrical Engineer
Hudson River Stone Corporation

AFTER exhaustive studies, the conclusion was reached by the officials of the Hudson River Stone Corporation that the erection of its own diesel-electric power plant to furnish the electric current needed for operation would effect a greater saving in power costs, thereby permitting a lower power cost per ton (or cubic yard) of stone crushed and handled, than would be possible by using purchased power. The cost of generating this power and supplying it throughout the system has been computed at approximately $0.0125 per kilowatt-hour for an 8- to 10-hr. day, assuming 4c. fuel oil and including all interest, maintenance, and depreciation charges. Should conditions call for operating 14 to 16 hr. per day, the kilowatt-hour cost would be materially lower.

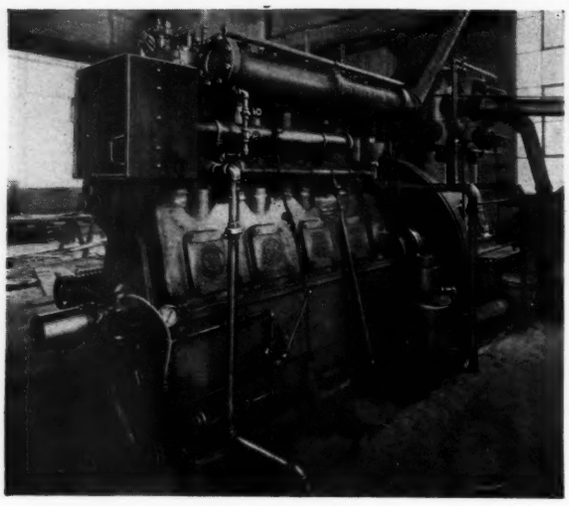
Selection of the generating units was so made as to give the greatest possible flexibility in plant operation. The prime movers are three six-cylinder, four-cycle solid-injection McIntosh & Seymour diesel engines, rated each at 660 b.hp. at 300 r.p.m. These were supplied, together with all auxiliaries except the water pumps and the cooling system, by the McIntosh & Seymour Corporation, Auburn, N. Y.

The generators are direct-connected to the diesel engines and have direct-connected exciters. They were furnished by the General Electric Company. Their rating is 625 kva. at 0.8 power factor (500 kw.) at 2,300 volts, 3 phase (Delta), 60 cycles. The exciters are rated 14 kw. at 125 volts, direct current.

For the control of the generator and


3-500-kw generators
150-hp. syn. motor
150-hp. syn. motor
Control Room, Bldg. No. 8
75-hp. slip-ring motor, conveyor No. 13
460-v. control
POWER PLANT
3-25-kva. transformers for station auxiliaries and dock control circuit
2,300-v. feeder
Conveyor 14
Conveyor 15
2,300-v. delivery feeder
8" sand pump, Bldg. No. 6
Transformer bank No. 1, 200-kva.
Crushing feeder No. 2
2300-v. feeder
Transformer bank No. 2 2-200-kva.
460-v. main distribution panel
Crushing feeder No. 1
Power distribution panels in Bldg. No. 3
Conveyor 12
Conveyor 11
Control Room, Bldg. No. 2 Conveyors 1, 2 and 3, revolving screens and 3-125-hp. gyratory motors
460-v. feeder for conveyors 1, 2 and 3 and 2 revolving screens
Conveyors 4 to 9 inclusive and 16 vibrating screens
Conveyor 10
460-v. control circuit for Bldgs No. 1 and 2 (2,300-v. motors)
Control Room, Bldg. No. 1 Pan feeder, jaw crusher and 2-30" gyratory crusher motors

Schematic diagram, Flow-sheet of electrical power distribution



Two hundred horsepower diesel-driven direct-current air compressor, installed in power plant

feeder circuits a manually operated switchboard has been provided. It comprises the following units:

One Tirrill regulator panel.
Three 500-kw. generator panels. One more to be installed in the future.
One totalizing panel (total output of generators).
Four 2,300-volt feeder panels.
One emergency light throw-over panel.
One 460-volt distribution panel.

The computed present load requirements are such that the three 500-kw. generator units will easily meet them. However, as soon as conditions call for operating the plant at full capacity, a fourth 500-kw. generator will be installed.

For operating the station auxiliaries a station service bank of three 25-kva. 2,300/460-volt single-phase transformers has been installed. A single 5-kva. transformer has also been provided for plant lighting, as well as an emergency storage battery with automatic throw-over for service in the event of failure of the supply of alternating-current power.

Means for taking power from the utility company's mains have also been provided so that the diesel cooling-water circulating system and the lubricating-oil heaters may be started cold or from "standstill."

The cooling-water circulating system of the diesel engines comprises the following, cited in order of operation:

One 7-tray cooling tower.
One 1,000-gal. surge tank.
One 10,000-gal. cold-water storage tank.
One 600-g.p.m. pump, 45-ft. head.
Two 1,500-gal. hot-well tanks.
One 600-g.p.m. pump, 90-ft. head.

The 600-g.p.m. 45-ft. head pump listed is for supplying cooling water to the diesel engines and the diesel-driven air compressor. The other pump, of a capacity of 600 g.p.m. at 90-ft. head, is on the "drain side" of the engines (the hot-well tanks acting as head or balancing tanks), and pumps the hot water from the diesel engines either through the heating coils in the main fuel-oil storage tanks and thence over the cooling tower, or, bypassing the fuel-oil tanks, directly over the cooling tower. Then the cycle is completed by passing the cooled water into the surge tank and thence by gravity to the 10,000-gal. storage tank.

Besides these two pumps other power-plant auxiliaries are also installed, as follows:

One small gasoline engine, 350 lb. starting air compressor (non-automatic).
One motor-driven 350 lb. starting air compressor (either full automatic or normal operation at will).
One bank of fuel-oil separator equipments for each diesel engine, each comprising
  One gear pump, motor-driven.
  One 5-kw. electric heater.
  One lubricating oil separator.
Two 5-kw. oil immersion electric heaters for each diesel-engine lubricating oil tank, to permit heating the lubricating oil in cold weather before starting the engines.
One 2½-kw. oil-immersion electric heater for each diesel-engine daily-storage fuel-oil tank, so as to provide for better fuel oil combustion.
One transfer pump for pumping fuel oil from the main storage tanks to the daily storage tanks in the power house.
Each fresh-air inlet of the diesel engines is provided with an American (Reed) air filter and metal louvres to permit removal of all possible dirt or foreign matter.
Three 350-lb. air-storage tanks for engine starting (one for each unit), all connected to a ccmmon manifold on both the intake and discharge sides so as to supply or draw air from any or all at will in accordance with the requirements.

In the power house a 200-hp. Chicago-Pneumatic direct-connected diesel-driven air compressor is also installed, and provision has been made for a second or future unit to furnish air for drilling operations.

The present main fuel-oil storage system consists of three 20,000-gal. tanks, with provision for the installation of a fourth tank when the additional diesel unit is installed in the future.

The piping arrangement is such that the tanks may be filled from tankers, tank cars, or trucks, and by proper arrangement of valves the tanks may be drained and oil pumped out to the tankers or tank cars.

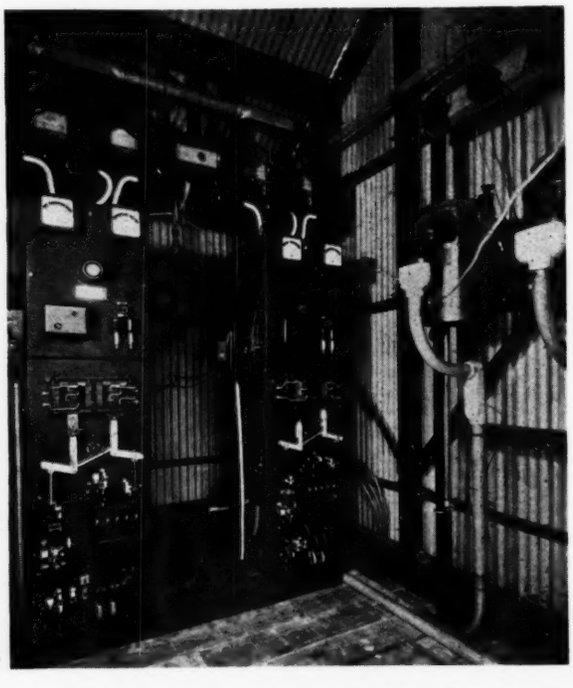
In the event of fire in the power plant, all daily oil-storage tanks can be "dumped" into a 1,000-gal. outdoor overflow tank.

All valves used in the oil lines are of the Merco-Nordstrom lubricated-plug type. This type of valve has also been installed in the crushing plant sand lines where standard gate valves could not be used because of the excessive wear caused by abrasive material, water and slimes.

The electrical power distribution system has four feeders. They include:

One 2,300-volt station service (power plant auxiliaries) feeder.
Two 2,300-volt crushing plant feeders (No. 1 and No. 2).
One 2,300-volt delivery feeder.

The first of these—namely, the station service feeder—supplies power to the three 25-kva. transformers for power plant auxiliaries. Crushing plant feeder No. 1 supplies 2,300-volt, 3-phase power



Control room for Building No. 8. Control equipment for two 150-hp. motors for water pumps.

to all of the 2,300-volt motors in buildings Nos. 1 and 2. Crushing plant feeder No. 2 supplies 2,300-volt, 3-phase power to the 75-hp., 2,300-volt slip-ring sand pump motor, and also to the two 2,300/460-volt, 3-phase, 200-kva., parallel-operated transformers (transformer bank No. 2). The latter transformers in turn supply all the 460-volt power for the 2,300-volt control equipment and for the 460-volt motors which drive the revolving and vibrating screens; also for the seven conveyors above the silos and those between buildings Nos. 1 and 2.

The delivery feeder supplies 2,300-volt power to the one 200-kw. 3-phase 2,300/460-volt transformer (bank No. 1), which, in turn, furnishes 460-volt power to all conveyor motors under the silos, and to the car- and truck-loading conveyors. A 2,300-volt sub-feeder is tapped off at transformer bank No. 1, which furnishes 2,300-volt power to the two 150-hp. 2,300-volt synchronous motors operating the 200-lb., 1,000-g.p.m. and the 100-lb., 2,000-g.p.m. water pumps on the dock as well as the 75-hp. slip-ring motor for No. 13 conveyor.

The control circuit for this equipment is a 460-volt feeder running from the power plant to the control transformer.

Owing to the dust conditions in this type of plant, the company decided to locate all the motor-control equipment for the motors in buildings Nos. 1 and 2 in separate control rooms. These rooms are made as dust-proof as possible. Each pushbutton control station for starting or stopping the motors is located at a vantage point of operation, which in many instances is adjacent to the motors. All of the control equipment for the vibrating screens and the conveyors over the silos in Building No. 3 are located in groups in the general vicinity of their respective motors. Because of the size of the building on the dock and the general arrangement of the equipment in it, construction of a separate house to inclose all of the control equipment as well as the control and lighting transformers was found necessary. Only a skilled attendant will have access to all of the control rooms and the transformer rooms.

All equipment and control for the motors and heaters is fully automatic pushbutton operated, which reduces and removes all possible hazard caused by unskilled laborers or operators.

Installation of an emergency lighting system throughout the entire plant is planned, whereby certain lights will be supplied continuously from the utility company's mains. The general illumination will be provided from the company's own power supply.

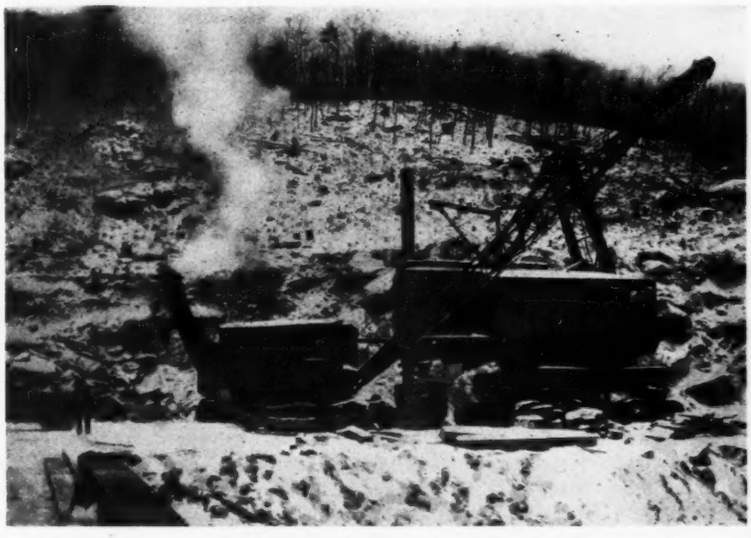
In addition to the power and light distribution system just described, a complete motor-interlocking and signal system is installed which makes it possible to shut down any equipment upon failure of the preceding unit.

Special attention is directed to the type of electric cables and wires used in this installation. All multi-conductor cables are varnished-cambric insulated and lead-covered. Over the lead covering the cable has a coating of heavy asphaltum paint and the entire lead cable is then protected with an interlocking steel-armor belt.

For the remote-control 460-volt circuits, both the three motor and the three control leads are combined in one cable, eliminating the necessity of installing an additional conduit or cable run. Selection of these types of cables was made because of the lower installation costs as compared with cable pulled in conduit. All cable and cable terminals were furnished by the General Cable Corporation, New York.

In view of the fact that the electrical installation had to be made in such a way that operations could be effective in five weeks from the start of work, the use of this cable made it possible to adhere to the original schedule, whereas if cable in conduit runs had been used, this could not have been possible, as all conduit would have had to be installed before any cable could have been pulled through it.

■



The quarry. The large 3½-yd. shovel, partly assembled, and the smaller 1½-yd. excavator. Also the crane.

■

Microscopical Determination Of Ore-Treatment Methods

Arthur S. Hecht

IN THE investigation of any ore, where the object is to plan a flow-sheet for its treatment, microscopical examinations will be found exceedingly helpful, either before or in conjunction with the regular physical-laboratory tests. Or, if the concentrating mill is already in operation, the use of the microscope will help to explain why certain results are obtained during concentration. Two types of microscopes may be used: the petrographic microscope and the binocular or ore-dressing microscope. Use of either will depend upon the problem at hand. If the genesis or the underground relation of one orebody to another is sought, the petrographic microscope is clearly the instrument to use. If, however, the object is to determine the milling characteristics of an ore, a choice may be made of either of the two instruments.

The problem discussed here pertains to the determination of the possibilities of treating the output of two operating mines of known chemical analyses so that a concentrated product to suit predetermined requirements may be obtained. For this investigation the petrographic microscope was selected, because thin sections, as well as natural products, were to be examined.

In the two mines mentioned in the foregoing the orebodies or lenses consist principally of two oxides of iron. One is magnetite, and the other martite, a non-magnetic iron sesquioxide under an isometric form. Strictly speaking, however, it is feebly magnetic, being attracted by a magnet of sufficient intensity. The non-magnetic term is used because it is more convenient and less confusing. Gangue material, consisting for the most part of quartz, the feldspar, mica, hornblende, and varying amounts of apatite, is associated with the iron.

The magnetite is of the massive type, and only occasionally does it show the well-formed crystal structure. On the other hand, the martite is always found in bodies of well-developed crystals which are pseudomorphous after magnetite. This feature, together with the respective behavior of the two minerals when tested with a hand magnet, readily serves to distinguish one from the other. Apatite, which is the source of the phosphorus in the ore, occurs with both forms of the iron. Its presence rarely can be detected with the naked eye or with an ordinary hand lens; hence the necessity of the microscope to study the occurrence of the apatite.

With a given set of conditions of crushing and screening already in operation in the mill, the following problems were to be studied:

1. The extent to which the valuable mineral grains were liberated.
2. The relationship between the valuable mineral and the apatite:
   (*a*) The manner in which the apatite occurred with both oxides of iron.
   (*b*) The size of the apatite grains.
3. The extent to which the apatite was liberated.

Table I—Screen Analysis of Ore Sample

Mesh	Weight, per Cent	Cumulative Weight, per Cent	Iron, per Cent	Phosphorous, per Cent
Minus 6 plus 8	12.8	12.8	53.16	1.082
Minus 8 plus 14	25.1	37.9	55.08	1.026
Minus 14 plus 28	19.6	57.5	55.56	1.156
Minus 28 plus 48	16.8	74.3	54.96	1.359
Minus 48 plus 100	12.1	86.4	54.00	1.812
Minus 100 plus 200	7.7	94.1	50.16	2.144
Minus 200	5.9	100.0	43.68	2.422
Average.....			53.70	1.379

The mill was already treating ore from No. 1 mine, and additional ore was to be supplied from the No. 2 mine. As a result of previous operations the latter was known to contain a higher percentage of phosphorus than the No. 1 ore, and hence an important point was to know whether it could be run with the No. 1 ore and under the same conditions of grinding and screening. For this purpose a 10-ton sample of the No. 2 ore was taken from all the exposed lenses in the mine. This was crushed and screened so that the product would duplicate as nearly as possible that produced in the mill. This completed, a second sample was carefully taken of the 10-ton lot. A screen analysis of the second sample is presented in Table I. From this master sample, three portions were taken, one of which was employed for each of three following purposes:

1. For a complete screen analysis of iron and phosphorus.
2. For a microscopical examination of each size to determine the amount of valuable mineral liberation.
3. For a microscopical examination to study the apatite as to size, distribution, physical relationship with the iron oxide, and the extent of liberation in the various sizes.

Table II—Mineral Classes and Weighted Value

Composition	Mineral Class per Cent	Weighted Value
All mineral.....	100	2.0
Mineral, 75 per cent; waste, 25 per cent.....	75	1.5
Mineral, 50 per cent; waste, 50 per cent.....	50	1.0
Mineral, 25 per cent; waste, 25 per cent.....	25	0.5
All waste.....	...	0.0

About 60 per cent of this ore was, by weight, magnetic. This would account for about 80 per cent of all the iron units present. I mention this fact now, before going on, because it will come up later when the relationship between the apatite and the iron oxides is discussed. A microscopical analysis was then made of another representative sample. The sample was screened, and the various sizes were examined separately. This yielded results of a quantitative and qualitative nature.

Each size was sampled after thoroughly mixing the grains on a sheet of paper. In each instance the product was poured through a funnel so that a small cone was formed on the paper. Taking one corner of the paper, the pile was given a rolling motion in one direction, care being taken that the grains did not slide on the paper but were rolled and mixed. A similar operation was performed with the opposite corner. The two remaining corners were employed in the same manner. By using two opposite corners at the same time, the grains formed a flattened cone, which was divided into four similar parts by two strips of metal set at right angles to each other. Two opposite quarters were selected and treated in the same manner until the required amount was obtained.

The plus 8 mesh and plus 14 mesh sizes were examined with a suitable hand lens rather than with the microscope, because with the latter the magnification was too large and proper focusing was not obtainable, owing to the irregularity of the grains. With the remaining sizes, from the minus 14 mesh downward, the microscope was used. This material was placed on regular petrographic slides, and the grains were examined with suitable magnification. The method employed with all the sizes was the same.

Each grain of the sample selected was examined and assigned to a certain class, as indicated in Table II. The weight for each class was used rather than the specific gravity, so that a unit value could be employed. This weight value is a number based on the specific gravity. Because the specific gravity

of magnetite or martite would be about twice that of the gangue material, a weight of two was used as the yardstick by which any intervening combination of mineral and waste could be weighed. A unit value of one might have been used, but the weighted values would have been more awkward fractions to deal with.

Table III can now be analyzed for both quantitative and qualitative results. Columns *A* and *B* show the screen test. Column *C* represents the actual number of grains examined in each size. Column *D* has each grain put in its proper class with its corresponding weight value, the total being shown in Column *E*. The theoretical all-mineral value, Column *F*, is arrived at by considering all the grains examined to be all-mineral or the 100 per cent class. This gives us a figure against which can be compared the actual total mineral value present in any given size. Dividing the actual by the theoretical value gives the mineral assay, Column *G*. From this it is simple to arrive at the metallic iron assay. The factor 0.719, used for this conversion, was determined as follows: 60 per cent of the sample was by weight magnetic. This accounts for about 80 per cent of the total iron units, which are present in the form of magnetite, or Fe_3O_4. The remaining 20 per cent is in the form of martite, or Fe_2O_3. The former, in the pure state, contains 72.4 per cent iron and the latter only 70 per cent. Combining the two, the conversion factor 0.719 results. The magnetic

Table III—Quantitative and Qualitative Results

A Mesh	*B* Weight per Cent	*C* Count	*D* Weighted Mineral, Value	*E* Total Value (*D*)	*F* Theoretical All-Mineral Value (C × 2.0)	*G* Mineral Assay, per Cent $\frac{E}{F}$	*H* Per Cent of Each Class Present in Each Size Classes 100	75	50	25	All Waste
Minus 6 plus 8	11.0	146	63 @ 2.0 21 @ 1.5 20 @ 1.0 22 @ 0.5 20 @ 0.0	188	292	64.5	44	14	14	14	14
			146 = 188								
Plus 14	25.6	311	180 @ 2.0 48 @ 1.5 37 @ 1.0 17 @ 0.5 49 @ 0.0	478	662	72.2	54	15	11	5	15
			331 = 478								
Plus 28	20.3	950	635 @ 2.0 40 @ 1.5 50 @ 1.0 225 @ 0.0	1380	1900	72.6	67	4	5	..	24
			950 = 1380								
Plus 48	17.5	638	401 @ 2.0 44 @ 1.5 39 @ 1.0 154 @ 0.0	907	1276	71.0 (a)	70	..	6	..	24
			638 = 907								
Plus 100	12.2	345	240 @ 2.0 105 @ 0.0	480	690	69.5	69	..	..	..	31
			345 = 480								
Plus 200	7.7	434	283 @ 2.0 151 @ 0.0	566	868	65.2	65	..	..	..	35
			434 = 566								
Minus 200	5.5	292	154 @ 2.0 138 @ 0.0	308	584	53.0	53	..	..	..	47
			292 = 308								

Weighted average mineral assay: 69.5. Iron assay = mineral assay × 0.719: 49.97. Chemical analysis iron assay: 54.45.

(a) The 75 per cent class was cast with the 100 per cent class for this figure because these grains were really between the two classes.

analysis, it can neverthless be relied upon, when the chemical analysis is not at hand, to give a fairly accurate determination of the mineral content. Such an examination provides the only means of obtaining a true mental picture of the distribution of the free, mixed,

Table IV—Distribution of Apatite in Ore Sample

	Minus 6 Plus 8 Mesh	Plus 14 Mesh	Plus 28 Mesh	Plus 48 Mesh	Plus 100 Mesh	Plus 200 Mesh
Apatite (microscopic), per cent	5.49	6.50	5.83	No determination made		
Phosphorus (microscopic), per cent	1.02	1.20	1.07	No determination made		
Phosphorus (chemical analysis), per cent	1.082	1.026	1.156	1.359	1.812	2.144
Apatite, included, per cent	35.5	28.6	15.0	0.0	0.0	0.0
Apatite, attached, per cent	58.8	53.5	14.0	17.0	0.0	0.0
Apatite, free, per cent	5.7	17.9	71.0	83.0	100.0	100.0

portion was determined on the Davis tube magnetic concentrator. Several valuable deductions can be drawn from the last column, *H*, which really describes what happens when the ore is crushed. These results may be listed as follows:

1. The largest increase in the production of free mineral grains occurs at a size between 14 and 28 mesh. This may be referred to as the critical size.
2. At this same size a marked decrease also takes place in the number of objectionable mixed grains, with the total disappearance of them somewhere between the 48- and 100-mesh sizes.
3. An idea is obtained as to the range in sizes where the most effective work can be done without the production of an unnecessary amount of fine material.

The first part of the microscopical examination, the results of which are tabulated in Table III, has served a double purpose. Although it was not meant to take the place of a chemical and waste grains in the crushed ore.

Finally, we come to that stage in the investigation in which a determination of the distribution and the occurrence of the apatite in the ore is to be made. Using a similar sample to that employed in the first part of the study, a screen test was made, and a representative sample taken from each size. Regular thin sections, for use with the petrographic microscope, were made of the individual sizes. The necessity for correct sampling, owing to the small quantity that can be used for the thin section, is of paramount importance; especially so for the larger grain sizes. And for this reason more than one section was made for each size, so that the duplicate results could be compared by the microscopist. Naturally, if the results of two sections of the same size were in harmony with each other, the sampling was assumed to be correct. Thin sections of the various sizes, rather than of the ore in its natural state, were used for the reasons listed in the following:

1. A fewer number of thin sections were necessary. This effected a saving of time, labor, and expense. Owing to the character of the ore, the investigation would have been difficult and the results doubtful had ore specimens of the original 10-ton sample been selected.
2. These sections told immediately and positively what would happen to the mineral, waste, and apatite grains when the ore was crushed. From a similar study of the ore specimens only a surmise could have been obtained.

The slide was placed on the stage of the microscope and each grain examined separately. Identification of the mineral was made, and the area noted. This latter figure was obtained with a suitable micrometer set in the eyepiece of the microscope. The average of the two measured axes was called the size of the grain. The total area of minerals for each slide at hand, the amount of apatite present, was calculated by taking into account the specific gravity of the latter and applying this to the total area of apatite. Whenever an apatite grain was spotted, its area was noted

Table V—Screen Analysis of Crushed Mill Feed

Mesh	Weight, per Cent	Cumulative Weight, per Cent	Iron	Phosphate
Minus 6 plus 8	12.5	12.5	45.45	0.756
Plus 14	28.8	41.3	53.58	0.739
Plus 28	22.5	63.8	54.48	0.852
Plus 48	15.8	79.6	51.33	1.269
Plus 100	9.7	89.3	47.60	1.651
Plus 200	5.7	95.0	43.43	1.912
Minus 200	5.0	100.0	37.56	2.146
Average:			50.46	1.072

in an appropriate column under the heading of included, attached, and free apatite. The results of this analysis, together with a comparison with the chemical analysis, are set forth in Table IV.

The size of the apatite grains varied considerably, ranging from 0.0005 in. or 100 mesh, and sometimes smaller, up to 0.10 in. or 8 mesh. Average size of the apatite was about 0.030 in. or 20 mesh. The largest increment in the liberation of the apatite takes place somewhere between minus 14 mesh and plus 28 mesh, with the absence of included or attached grains on the plus 100 mesh. The apatite is associated in its natural state with the iron oxides. It is disseminated throughout the massive magnetite, whereas, when it occurs with the crystalline martite, it is mainly found between the individual crystals rather than within the martite itself. Hand specimens of large martite crystals, which could be selected almost at will, would assay very low in phosphorus, whereas with the magnetite this would be impossible. As far as the magnetic concentration is concerned, a product finer than that shown in Table I would have to be delivered to the magnetic drums if the phosphorus were to be removed with greater success.

About a year later a sample of the crushed mill feed was taken. This was made up of the No. 1 and No. 2 ores.

Table VI—Microscopical Determination of Distribution of Apatite in Crude Ore

	No. 2 Ore Sample: Iron, 53.70; Phosphorus, 1.379		Mill Feed, No. 1 and No. 2 Ore Samples: Iron, 50.46; Phosphorus, 1.078	
Minus 6 plus 8 mesh:	Per Cent in Each Size	Per Cent of Total Apatite	Per Cent in Each Size	Per Cent of Total Apatite
Included apatite	35.5	3.6	41.2	3.6
Attached apatite	58.8	5.9	58.8	5.2
Free apatite	5.7	0.6	0.0	0.0
Plus 14 mesh:				
Included apatite	28.6	5.4	14.5	2.9
Attached apatite	53.5	10.0	53.2	10.5
Free apatite	17.9	3.3	32.3	6.4
Plus 28 mesh:				
Included apatite	15.0	2.5	1.7	0.3
Attached apatite	14.0	2.3	36.2	6.5
Free apatite	71.0	11.7	62.1	11.1
Plus 48 mesh:				
Included apatite	0.0	0.0	1.5	0.3
Attached apatite	17.0	2.8	49.5	9.2
Free apatite	83.0	13.8	49.0	9.1
Minus 48 mesh; plus 100 mesh; plus 200 mesh; minus 200 mesh:				
Free apatite	100.0	38.1	100.0	35.0
Total free apatite, per cent	67.5		61.5	

A screen analysis, presented in Table V, shows the material to be almost the same as that combined in the laboratory, a screen analysis of which is shown in Table I. Thin sections of the various sizes were prepared from this sample taken a year later, and a microscopical examination was made.

The results obtained on both samples are similar, but only up to a certain point. Percentage of the total apatite in each class is about the same. However, the amount of free apatite is different for each class. This variance was accounted for by the presence in the mill-feed sample of a greater amount of 60- or 80-mesh apatite grains, inasmuch as the amount of liberated apatite decreased in the 48-mesh group.

The value and trustworthiness of the conclusions reached by the use of a microscope can, in the final analysis, be best judged by actual concentration tests. Where the mill is already in operation, these tests can easily be made. But when no such facilities or milling records are available, the conclusions so reached should be relied upon to aid in the planning of such tests or in the design of a flowsheet.

■

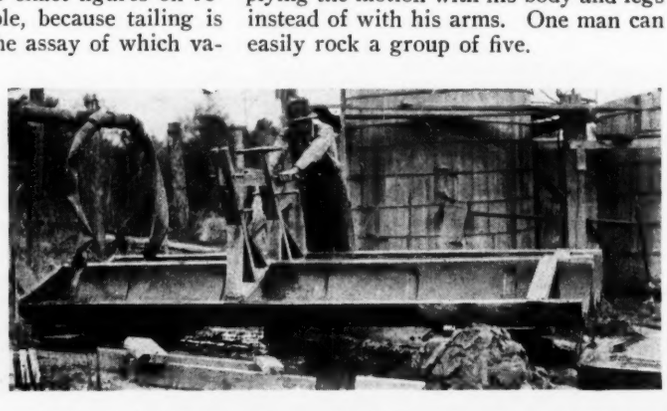
Wooden Rocker Recovers Fine Gold

A SIMPLE yet efficient type of twin-unit gold-recovery apparatus is illustrated herewith, details of the design and operation of which have been kindly supplied by Mr. T. W. Swoyer, of the Black Ankle Mining Corporation, Seagrove, N. C., where it is at work.

The rockers, constructed from a solid poplar log, are 14 ft. long. Sometimes they are made shorter, but rarely longer. The width is 14 in. and the depth 8 in. The trough, slightly elevated (4 in.) at the discharge end, has five depressions. Of these, the first two, or those nearest the feed, are 2 in. deep and 6 in. wide; the lower three are 1 in. deep and 5 in. wide. The units, hand rocked at 50 to 60 strokes per minute, are mounted on timbers from which a flat, iron pin protrudes, fitting into a slot in the bottom of the rocker. This keeps them in place and avoids one of them striking the other when in motion. The natural roundness of the log provides the rolling surface, and in this respect indicates difference from the old "cradle."

The feed now being sent to them is a little too coarse—from 10 to 30 mesh, but the intention is to grind to minus 30 mesh, although this may cause sliming. Balls are not usually employed except to break up clay and prevent caking in the bottom. As soon as this clay has been broken up, they are removed, as they tend to "flour" the quicksilver, which may then reach the lower grooves. Mercury is fed only in the two upper grooves, in which the amalgamation takes place, although in this installation it starts at the head of the sluice from the mill, which is about 200 ft. long and lined with burlap sacking. Free gold that passes the burlap—also treated with mercury for 12 ft. below the discharge of the mill—is caught in the rockers, as well as any amalgam washed down. No exact figures on recovery are available, because tailing is being re-treated, the assay of which varies considerably. Panning at the discharge end of the rockers, however, has always failed to show any gold, and the losses are only in the form of a little flour quicksilver.

Capacity depends on the size of the feed. The units illustrated treat about 10 tons per ten-hour day. The burlap sacking and the rockers catch a large amount of sulphides, later concentrated or floated. The rockers were built by Tom Morgan, an expert in their construction. They are sometimes used in batteries of five or ten, rocked by a man standing on a platform above, and supplying the motion with his body and legs instead of with his arms. One man can easily rock a group of five.



Poplar-wood rockers, sloping upward toward the discharge, prove effective commercial panners

Cost of Surface Metal Mining

■

Morris J. Elsing

■

SURFACE MINING is known by a variety of similar terms, such as open-pit mining, open-cut mining, and quarry mining. Glory-hole or mill-hole mining may be either entirely underground or it may be a combination of underground and surface work. The surface method consists of primary breaking into surface mill holes, with secondary breaking and handling underground. Most forms of placer mining are surface work, but that subject will not be discussed in this article.

According to the U. S. Bureau of Mines, 42 per cent of all ore produced by the important metal mines in the United States in 1929 came from surface operations. The copper mines produced by surface methods 31 million tons, or 49 per cent of the total copper ore; surface iron mines produced 40 million tons, or 51 per cent of all the iron ore.

Table I—Mining Costs, Including Stripping Absorbed, Utah Copper Company. From Annual Reports

	Cents per Ton				
	Direct Mining	Overhead and Fixed	Total Mining	Stripping Absorbed	Total Cost
1922	18.7	5.3	24.0	12.5	36.5
1923	19.2	3.2	22.4	12.5	34.9
1924	19.6	4.0	23.6	12.5	36.1
1925	17.1	4.1	21.2	12.5	33.7
1926	15.2	3.7	18.9	12.5	31.4
1927	12.9	5.1	18.0	19.3	37.3
1928	11.7	3.9	15.6	16.2	31.8
1929	12.4	6.4	18.8	22.4	41.2
1930	13.2	13.1	26.3	24.1	50.4
1931	12.4	6.1	18.5	21.0	39.5

Surface mining usually includes three primary operations: breaking, loading and transportation of the ore as well as the waste that usually overlies the ore to be mined. In some countries, where the wages amount to only a few cents a day, very low costs of mining result by the use, almost entirely, of hand methods. Under other conditions, scrapers and teams are employed to load wagons or cars. Almost the entire American tonnage above mentioned was produced by highly mechanized methods.

In the Western United States, six copper mines have employed open-cut methods of mining—Utah Copper, New Cornelia, Nevada Consolidated, Chino Copper, United Verde, and Copper Queen. Each of these is herewith described, and the most important costs are presented.

The Utah Copper Company, Bingham Canyon, Utah, has mined more than 200,000,000 tons of ore from its open-pit mine at Bingham Canyon. In mining this ore, about 220,000,000 tons of waste stripping has been removed, the ratio of ore to waste being 1 ton of ore to 1.1 tons of waste. The orebody is 6,000 ft. long, 4,000 ft. wide, and is 2,000 ft. thick. An average of 115 ft. of leached overburden above the ore is almost barren of copper. The ore consists of mineralized monzonite porphyry, which has been secondarily enriched, raising the grade to about 1 per cent copper. The monzonite is well fractured and crushed, so that primary blasting is effective in breaking it sufficiently small for economical handling by the shovels.

In June, 1907, the first steam shovel started working on ore. Since then many changes in operating methods and equipment have been made. The present practice is to make benches from 40 to 50 ft. high. Ore is broken by drilling and blasting toe holes, 22 ft. deep, dipping slightly downward. Holes, 15 ft. apart horizontally, are drilled with the old-style 3½-in. piston drills, mounted on tripods. Holes start at 3¾ in. in diameter and end at 2½ in. The hole is sprung four times, with 7, 15, 30, and 50 sticks of powder before the main blasting, which is done with 150 to 250 lb. of low-freezing ammonium nitrate powder. Boulders are blasted with "dobe" plasters, or by blockholing.

The average footage drilled per shift is 45 ft., or at the rate of 0.022 ft. per ton of ore broken. Eight tons of ore is broken per pound of powder. The cost for explosives is 1.5c. per ton of ore. Total cost of drilling and blasting—including labor, equipment, maintenance, pipe lines, and explosives—amounts to less than 3c. per ton.

Electric shovels mounted on caterpillar tractors are now used. Dipper capacity is 45 cu.yd., and the average output per shovel-shift for 1928 was 3,843 tons of waste or ore. Ore trains, made up of ten to twelve 80-ton ore cars, are loaded in 1¼ to 1½ hr. Waste trains are made up of from three to seven 30-cu.yd. cars. Electric-trolley locomotives are replacing the older steam locomotives.

Normal output per day is 60,000 tons of ore and 30,000 cu.yd. of waste. The ore is transported 18 miles to the Arthur and Magna concentrators at Magna. Concentrates are shipped 4 miles to the Garfield smelter. From 1904 to 1931 inclusive, 210,432,338 tons of ore was produced. Mining costs for the past ten years are shown in Table I.

Direct mining costs are made up of the items as shown in Table II, taken from U. S. Bureau of Mines Information Circular 6234. Costs are for 1928. The item "Transportation" includes only pit transportation and not transportation to the concentrators.

Up to Jan. 1, 1931, the New Cornelia Branch of the Phelps Dodge Corporation, Ajo, Ariz., produced 32,400,000 tons of copper ore by the steam-shovel method of mining. Of this, 16,800,000 tons was carbonate ore, treated in a leaching plant; and 15,600,000 tons was sulphide ore, treated in a concentrator. The mining of this ore required the removal of 6,800,000 tons of waste, a large part of which was not stripping but material which occurred in the transportation approaches to the pit. To date, 0.21 ton waste has been removed per ton of ore mined. In mining the remaining ore, about 0.8 ton waste will be removed per ton of ore to be mined. The pit is 3,600 ft. long and 2,500 ft. wide.

The orebody is composed of monzonite and diorite. Average thickness of the ore is 425 ft., of which 55 ft. was carbonate ore, all of which has now been mined. The ore, on the average very hard, breaks into large boulders, on account of the presence of widely spaced fracture planes throughout the rock.

The most economical height for benches has been found to be 25 ft. Ore is broken by drilling vertical holes to a depth of about 2 ft. below the new bench level. An average blast consists

Table II—Direct Mining Costs 1928. Utah Copper Company

	Cents Per Ton. Ore	*Cents Per Cu Yd* Waste
Operating labor	3.8	11.4
Air	0.2	0.3
Explosives	1.5	3.4
Other supplies	0.5	1.6
Repair, labor and supplies	2.0	4.4
Power	0.3	0.7
Bench transmission lines	0.1	0.2
Fuel	0.7	1.7
Yard and trains	0.8	....
Ore cars	0.1	1.0
Supervision	0.5	1.3
Churn drilling	0.6	....
General	0.6	1.3
Totals	11.7	27.3
(Recapitulation)		
Drilling and blasting	2.7	6.0
Loading	1.9	4.1
Transportation	3.7	7.5
Track maintenance	1.7	7.1
Other mine charges	1.7	2.6
Totals	11.7	27.3

This article is one of a series on the cost of metal mining in the United States.

Table III—Operating Costs, New Cornelia, Ariz., per Dry Ton Ore and Waste Mined in 1930

	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)
Drill and blast	$0.035	$0.003	$0.001	$0.003	$0.001	$0.025	$0.002	$0.070
Loading	0.026	0.001	.....	0.001	0.013	.....	0.008	0.049
Transportation	0.042	0.001	.....	.....	0.016	.....	0.007	0.066
Miscellaneous	0.001	0.016	.....	.....	.....	.....	0.008	0.025
Totals	$0.104	$0.021	$0.001	$0.004	$0.030	$0.025	$0.025	$0.210

(1), Labor, (2) Supervision, (3) Compressed air, drills, and steel, (4) Electric power, (5) Fuel, (6) Explosives, (7) other supplies, (8) Total.

	Dry Tons
Ore loaded	2,376,764
Waste loaded	1,040,610
Total	3,417,374

of about 100 holes drilled in an area about 225 ft. long and 90 to 100 ft. wide. Holes are staggered and are laid out at 15-ft. intervals. Drilling is done by hammer drills suspended from 18-ft. pipe tripods. Drill steel is hollow-round and 1¼ in. diameter. The gage of the starter is 2 7/16 in. A drill crew consists of two men, who drill on the average 82.5 ft. of hole per shift. Where necessary, toe holes are also drilled.

All holes are sprung with 1⅛-in. 50 per cent gelatin powder, and the main charge in dry ground is 40 per cent granular powder. In wet ground, 40 per cent gelatin stick powder is used.

Secondary blasting is done by one jackhammer man assigned to each shovel. Tons of ore and waste broken per pound of powder in primary and secondary blasting is given as follows: Year 1921, 3.05 tons; 1922, 3.27; 1923, 2.67; 1924, 2.27; 1925, 3.37; 1926, 3.93; 1927, 4.22; 1928, 4.39; 1929, 5.85; 1930, 5.93.

Improved blasting practice and lower benches account for the great increase in tons broken per pound of powder shown in recent years.

Ore and waste are loaded by Osgood railroad-type steam shovels fired with oil. Dippers have a capacity of 4 cu.yd. Cars with a capacity of 20 cu.yd. are equipped with automatic air-dumping devices. A train consists of five or six cars, each of which has a capacity of about 40 tons. The haul is about 1.7 miles. In 1930 the average output per shovel-shift was 1,322 tons of waste or ore. The average daily production of sulphide ore since July, 1930, has been about 6,500 tons.

The foregoing information and the costs in Table III have been taken from U. S. Bureau of Mines Information Circular 6666.

On this basis, the direct cost of mining in 1930 was 21c. per ton; waste stripping adds 9.2c. per ton of ore, making a total of 30.2c. per ton of ore. Comparative open-pit copper mine costs are shown in Table X and certain statistical data in Table XI.

The main orebody of the mine of the Nevada Consolidated Copper Company, at Ruth, Nev., occurs in a monzonite porphyry, which has intruded Paleozoic limestones and shales. Parts of this monzonite have been secondarily enriched, forming an orebody running from 1.0 to 1.5 per cent copper. The main open-cut orebody is 4,000 ft. long by 2,500 ft. wide and up to 450 ft. thick. The amount of capping is irregular and varies from 20 ft. to 150 ft.

Vertical height of benches is 50 ft. Drilling is done with 6-in. churn drills, holes being spaced 35 ft. apart, 10 to 12 ft. back from the edge of the cut, and drilled to a depth of 55 ft. Before blasting they are sprung one or more times. Bucyrus caterpillar shovels load 55-ton Ingolsby ore cars. The average output of shovels is 2,759 tons of ore per shift, or 1,128 cu.yd. of stripping. The normal production of the mine is 18,000 tons of ore and 14,000 tons of waste per day. Ore is shipped to the company's Steptoe flotation mill and smelter at McGill, which is 22 miles from the mines. Costs are shown in Tables IV and V.

The ore at the Chino open pit of the Chino Copper Company, Santa Rita, N. M., occurs in a rectangular mass 5,000 ft. long and 4,000 ft. wide. Somewhat off center within this area is a core of barren granodiorite, 2,500 ft. long and 1,500 ft. wide. Ore occurs in diorite porphyry and sedimentary rocks, and in granodiorite that surrounds the central core. Alteration has been intense, resulting in the formation of silicified areas where the ore is hard and tough, and in soft areas where the ore contains much kaolin and sericite, with abundant colloidal matter. Overburden is in some places absent, and from this condition it varies up to 150 ft. thick, whereas much of the orebody is from 300 ft. to 600 ft. thick. The ratio of ore to stripping is 1 ton ore to 1.5 tons waste.

Benches are 50 ft. high. Bench drilling is done with gasoline churn drills with holes 4½ to 6 in. diameter. Holes are spaced 20 to 30 ft. apart and 10 ft. back from the edge of the bank, and drilled 5 ft. below the floor of the cut. Toe holes, 15 ft. apart and 22 ft. deep, are drilled to a depth of 5 ft. below the floor level of the cut. All holes are sprung. Small jackhammers are used for blockholing.

Electric and steam shovels are used. Shovel output has increased from 825 cu.yd. in 1923 to 1,280 cu.yd. per shovel-shift, or from 1,650 to 2,560 tons. Ore and waste cars of from 20 to 30 cu.yd. capacity are used in trains of 200 cu.yd. The average haul for ore is 2.7 miles; for waste, 2.66 miles. Ore passes through a primary crusher and is then delivered to the Santa Fe R.R., by which

Table IV—Mining Costs, Including Stripping Absorbed, Nevada Consolidated Copper Company. From Annual Reports

	Cents per Ton of Ore		
Year	Direct Mining	Stripping Absorbed	Total Mining
1922	22.2	40.0	62.2
1923	20.2	40.0	60.2
1924	21.4	40.0	61.4
1925	20.1	40.0	60.1
1926	19.0	40.0	59.0
1927	19.7	40.0	59.7
1928	22.4	40.0	62.4
1929	24.2	40.0	64.2
1930	28.2	40.0	68.2
1931	26.6	30.0	56.6

Table V—Nevada Consolidated Direct Mining Costs

	1920	1921	1922	1923	1924
	Cents per Ton of Ore Produced				
Drilling and blasting	6.2	6.4	3.8	4.2	3.9
Steam shoveling	3.0	2.8	2.7	4.7	4.4
Locomotive tramming	8.6	7.7	4.4	6.8	8.3
General pit expense	2.5	1.8	3.7	1.7	1.4
Administrative and general	9.8	9.4	7.6	2.8	3.4
Totals	30.1	28.1	22.2	21.4	21.4
	Cents per Cubic Yard of Stripping Removed				
Drilling and blasting	8.8	11.3	9.5	9.4	7.7
Steam shoveling	5.9	5.8	6.2	11.9	9.2
Locomotive tramming	18.7	18.2	16.3	20.6	23.2
General pit expense	1.0	1.2	2.6	2.6	2.0
Administrative and general	6.1	4.7	8.8	5.3	5.6
Totals	40.5	41.2	43.4	49.8	47.7

Table VI—Mining Costs, Including Stripping Absorbed. Chino Copper Company. From Annual Reports

	Cents per Ton of Ore		
Year	Direct Mining	Stripping Absorbed	Total Mining
1926	36.4	45.0	81.4
1927	34.8	45.0	79.8
1928	32.2	45.0	77.2
1929	31.4	45.0	76.4
1930	33.4	45.0	78.4
1931	26.7	35.0	61.7

it is hauled to Hurley, a distance of 10 miles, where the concentrator is situated. The concentrates are sent to El Paso, where they are smelted.

Up to the end of 1931, the Chino mine had mined by power-shovel methods 51,163,737 tons of ore. Mining costs are shown in Tables V and VI.

The West orebody of the Copper Queen mine of the Phelps Dodge Corporation at Bisbee, Ariz., occurred as a secondarily enriched pyritic ore containing some chalcopyrite and bornite in a siliceous breccia and porphyry. Its maximum length and width were 1,200 ft. and 800 ft. respectively. At a depth of 300 ft. these dimensions decreased to 500 ft. by 300 ft. The ore contained hard and soft ribs and the capping was generally hard. After due consideration of all underground mining methods, the open-pit method with steam shoveling was adopted.

Benches were 30, 45, and 60 ft. high. Bank holes, 10 ft. back from the edge, were drilled with churn drills and toe holes with Burley air drills, drilling an average of 27 ft. per shift. Bucyrus shovels, with 3.5-cu.yd. dippers, were used, and Porter locomotives hauled Western wheeled scraper cars of 20- to 25-cu.yd. capacity.

Open-pit mining with power shovels commenced at the United Verde mine of the United Verde Copper Company, at Jerome, Ariz., in 1918. To Jan. 1, 1929, about 8,000,000 cu.yd. stripping and 1,500,000 cu.yd. ore had been removed. Two distinct methods of open-pit mining have been employed, distinguished as the upper-pit method and the lower-pit method. The former involved the use of large steam shovels loading into railroad cars; the latter method involved the use of small electric shovels loading into auto trucks, which delivered ore and waste to mine raises from the main tunnel level.

In the upper pit, breaking was done by churn drilling bank holes and blasting with ten to twelve boxes of gelatin. In this way the altered diorite was removed, but when the fresh, hard, dense, unaltered diorite was encountered this method of drilling was not satisfactory. A combination of toe and bank holes drilled with machine drills proved the most desirable. The United Verde open pit is in harder rock than any of the other large copper mines in the Southwest, and this is reflected in the cost of breaking, shown in Table VIIIa.

Coyote blasting has also been done, and a total of 893,000 cu.yd. material was broken in this way at a total cost of $0.172 per cubic yard, of which $0.073 per cubic yard was for development, including labor and supplies, and $0.099 was for explosives. The powder consumption was 0.966 lb. per cubic yard.

All work in both the upper and lower pits was in an area of old stopes in which a mine fire has been burning for several years. Very hot ground necessitated the greatest care in loading holes and blasting. A charge of $1.50 per ton of ore is made in order to absorb the cost of stripping. Costs are shown in Table IX.

Of the many smaller Western open-cut mines, several are interesting from the standpoint of the cost of production. The New Idria Quicksilver Mines, Inc., San Benito, Calif. (I. C. 6462), in 1930 was producing at the rate of about 20,000 tons of ore per month from an open-cut mine. The broken ore is loaded into auto trucks with gasoline shovels mounted on tractors. Total direct mining cost for September, 1930, was 32c. per ton of ore. Stripping costs were 21.2c. per ton of ore, and auto-truck transportation to ore chutes cost was 5c. per ton, making a total of 58.2c. per ton.

The method of mining the waste stripping is unique in metal mining. The waste is broken into a mill hole and transported through a tunnel to the waste dump by one of the largest belt conveyor installations in the United States. About 1,500,000 tons has already been removed. The total direct cost of breaking and delivery of the waste to the dump is 10c. per ton. The length of the conveyor is 1,250 ft. The material is easily broken by drilling and blasting; but, in spite of this fact, the cost mentioned in the foregoing is truly remarkable, being one of the lowest breaking and handling costs of which there is record.

Mountain Copper Company, Shasta, Calif. (*E.&M.J.*, June 22, 1931), is producing 550 tons of gold ore per day from an open pit. Electric shovels load 3.5-ton auto trucks, which deliver the ore to the mill. The total direct cost of mining, including local administration

Table VII—Direct Mining Costs Steam and Electric Shovels

(1929 — I. C. 6412)

	Cents per Ton Ore		*Cents per Cu. Yd. Waste*		*Cents per Cu. Yd. Both*	
	Electric Shovels	Steam Shovels	Electric Shovels	Steam Shovels	Electric Shovels	Steam Shovels
Drilling and blasting	4.9	5.1	9.7	10.7	9.7	10.6
Shoveling	3.6	7.9	7.8	19.3	7.6	18.2
Haulage	10.8	13.1	21.9	28.4	21.8	27.7
General	1.4	1.7	3.0	4.9	2.9	4.5
Totals	20.7	27.8	42.4	63.3	42.0	61.0

Table VIII—Copper Queen Average Yearly Direct Cost per Solid Yard

(A.I.M.E. Vol. 68)

	Cents per Cubic Yard			
	1918	1919	1920	1921
Breaking ground	35.7	23.9	21.4	17.3
Steam shoveling	23.6	18.1	15.7	13.0
Haulage expense	33.3	23.9	25.4	17.7
Dump expense	4.0	3.1	3.1	2.3
Miscellaneous	0.8	0.8	2.7	1.6
Monthly labor bonus	...	0.4	2.3	2.9
Totals	97.4	70.2	70.6	54.8
Cost per ton (estimated)	45.1	32.5	32.7	25.4

Table VIIIa—Comparison of Costs of United Verde Drilling and Blasting Methods

	Per Cubic Yard of Material		
	Churn Drill (Soft)	Toe Hole	Machine Drilled Toe and Bank Hole (Hard)
Labor	$0.047	$0.063	$0.113
Explosives	0.026	0.044	0.069
	$0.073	$0.107	$0.182
Output per shovel-shift	800	530	758
Labor cost per foot of hole	$1.97	$0.22	$0.42

Table IX—Direct Cost, United Verde Open Pit

(I. C. 6248)

	Upper Pit		*Lower Pit*		
	Cents per Ton Ore 1925	Cents per Cu.Yd. Stripping, 1925	Cents per Ton Ore, 1928	Cents per Cu.Yd. Ore and Stripping 1928	Cents per Cu.Yd. Stripping, 1928
Drilling and blasting	14.9	24.1	13.9	32.7	29.3
Loading	11.8	21.4	7.7	18.0	16.1
Transportation (a)	10.3	21.1	8.6	18.7	16.0
Dump maintenance	....	5.8	....	....	3.8
Miscellaneous	....	....	....	....	1.8
	37.0	72.4	30.2	69.4	67.0

Steam Shovel Stripping Cost in 1925

(M.C.J, April, 1927)

	1925 Cents per Cu.Yd. Steam Shovel	1926 Cents per Cu.Yd. Electric Shovel
Machine drilling and blasting	21.6	....
Churn drilling and blasting	2.4	....
Steam shovel operation	21.3	18.4
Locomotive tramming and disposal	26.0	(b) 18.2
	71.3	....

(a) Locomotive transportation was in use in the upper pit, whereas auto-truck tramming was used in the ower pit. (b) Auto-truck tramming.

Table X—Comparative Open-Pit Copper Mine Costs

Direct Operating Costs

Costs per Ton Year	Utah 1928	Nev. 1924	Chino 1929 (1)	Chino 1929 (2)	U. V. 1925 (3)	U. V. 1928 (4)	C. Q. 1921	N. C' 1930
Drilling and blasting	$0.027	$0.039	$0.049	$0.051	$0.149	$0.139	$0.080	$0.070
Power shoveling	0.019	0.044	0.036	0.079	0.118	0.077	0.061	0.049
Locomotive tramming	0.037	0.083	0.108	0.131	0.103	0.086	0.082	0.066
General pit expense	0.017	0.014	0.014	0.017	.....	.....	0.010	0.025
Administrative and general	0.017	0.034	.....	.....	.....	.....	0.021	.....
Total direct	$0.117	$0.214	$0.207	$0.278	$0.370	$0.302	$0.254	$0.210
Stripping absorbed	0.162	0.400	0.450	0.450	1.500	1.500	0.711	0.092
	$0.318 (5) (7)	$0.614	$0.657	$0.728	$1.870	$1.802 (6)	$0.965 (8)	$0.302 (9)

(1) Electric shovels, (2) Steam shovels, (3) Upper pit, (4) Lower pit, (5) Electric locomotives, (6) Auto-truck tramming, (7) Overhead and fixed, 3.9c. per ton included, (8) Estimated from data on solid yards, (9) Cost of stripping prorated to tons mined.

Cost per Cu. Yd. Stripping	Utah 1928	Nev. 1924	Chino 1929 (1)	Chino 1929 (2)	U. V. 1925 (3)	U. V. 1928 (4)	C. Q. 1921
Drilling and blasting	$0.060	$0.077	$0.097	$0.107	$0.241	$0.293	$0.173
Shoveling	0.042	0.092	0.078	0.193	0.214	0.161	0.130
Locomotive tramming	0.075	0.232	0.219	0.284	0.211	0.160(5)	0.177
General pit expense	0.071	0.020	0.030	0.049	0.058	0.056	0.068
Administrative and general	0.026	0.056	.....	.....	.....	.....	.....
Totals	$0.274	$0.477	$0.424	$0.633	$0.724	$0.670	$0.548

(1) Electric shovels, (2) Steam shovels, (3) Upper pit, (4) Lower pit, (5) Auto-truck tramming.

comparatively soft but sufficiently hard so that the underground workings, including tunnels and raises, will stand without timber.

Quarry costs, excluding development, were 36.5c. per ton. Transportation from the quarry to the mill cost 10.8c., making a total cost of 47.3c. per ton. By the glory-hole method, costs were reduced to 22.6c. per ton. Of this amount, development cost 3.7c.; ore breaking, 14.3c.; loading and tramming, 3.4c.; and miscellaneous costs were 1.2c. per ton. Production was at the rate of 3,500 tons. The Fresnillo cost is one of the lowest which modern records present.

Of all the factors that determine the cost of surface mining, the ratio of the amount of ore to the amount of stripping to be removed is the most important. Obviously, with excessive amounts of overburden, surface operations must give way to underground methods. In all of the power-shovel operations at Western copper mines, the cost of stripping per ton of ore is considerably more than the direct mining cost. Utah Copper's cost of stripping is about one-half of the total mining cost per ton. At Nevada and Chino, stripping costs per ton of ore are twice as much as the direct mining costs. The record for United Verde shows that prepaid stripping adds $1.50 per ton of ore mined, whereas the direct mining cost is less than 40c. per ton. Stripping costs were also high at Copper Queen.

The physical character of the ground to be broken has an important bearing on the direct cost of mining. This is reflected in the method of drilling and the amount of footage required to break the ore or waste. Character of the ground determines the amount and kind of explosives required in primary break-

expenses, is 25c. per ton. Trucking to the mill is done for 13.6c. per ton, the total cost of ore delivered to the mill being 38.6c. per ton.

During 1927 and 1928 the Flowery mine, Virginia City, Nev. (*E.&M.J.*, Oct. 20, 1928), was producing 200 tons per day from an open pit by the glory-hole method. The direct cost of mining for the two-year period was 37c. per ton of material mined and removed. Total direct cost of ore delivered to the mill was 53c. per ton, which includes the cost of waste stripping.

Of the costs of the many famous old mines, but two are mentioned. Wasp No. 2, in the Black Hills of South Dakota, had a total mining cost of 54c. per ton in 1911. In the early days of the historic Treadwell group, the surface ores were mined by the glory-hole method at a cost of slightly less than $1 per ton.

Mining methods and practice in the Iron Country open-pit mines are not radically different from those of the Western copper mines. The old railroad type of shovel has been replaced by caterpillar shovels of large capacity. In the most efficient operations, shovels are operated by electricity and transportation facilities have been electrified. The technical press and company reports have given much more publicity to the costs of Western power-shovel mines than to those of similar operations in the Iron Country of Minnesota and Michigan.

Detailed comparative costs are not available. According to W. L. Taylor (*M.C.J.*, Oct., 1929), the average open-pit costs in 1928 on the Mesabi range, reduced to the short-ton basis, were as shown in Table XII.

Although this article is essentially a discussion of the cost of mining in the United States, two of the most famous foreign operations are mentioned: Chile Copper because of its size and importance as a copper producer, and Fresnillo, in the State of Zacatecas, Mexico, because of its remarkably low costs. Unfortunately, detailed costs of operations are not available for the Chile Copper operations. Annual reports show that the total operating costs per ton of ore for 1925 to 1928 are $1.45, $1.46, $1.72, and $1.52, respectively. These include ore treatment and all other operating costs, and exclude only interest, taxes, and depreciation. Electrolytic copper is produced, as wirebars and cakes. Operations are modern in every respect. Shovels, electrically operated, are mounted on caterpillar tractors. Electric locomotives of different types haul the trains. Ore-bearing costs (*Min. & Met.*, Dec., 1925) are somewhat higher than at most of the other large power-shovel mines. In one blast of 800,000 tons, coyote tunnels and drill holes cost 4.3c. per ton of material broken. Explosives cost 5.4c., with a total cost for primary breaking of 11.3c. per ton. A very small amount of secondary breaking is done. In point of amount of tonnage handled per year, Chile ranks second to Utah Copper. Excluding Utah, its production is more than that of the combined output of any two other power-shovel operations in the United States.

At the Fresnillo mine (*E.&M.J.* Dec. 1, 1923), between 1920 and 1923, the quarry method of mining was changed to a glory-hole system. The ore is a fractured graywacke sediment, which is

Table XI—Statistical Data, Power Shovels

	Year	Tons Ore Mined	Ratio of Stripping to Ore	Cost of Explosives per Ton Ore, Cents	Cost of Breaking per Ton Ore, Cents
Utah	1928	16,558,500	0.5 to 1	1.5	2.7
Chile	1928	9,466,985	.....	5.4	11.3
Chino (electric)	1929	2,621,340	1.8 to 1	2.1	4.9
Chino (steam)	1929	1,335,476	1.8 to 1	2.3	5.1
New Cornelia	1930	2,376,764	0.4 to 1	2.5	7.0
Nevada	1924	2,723,719	1 to 1	...	3.9
United Verde	1925	272,005	2.4 to 1	4.6	14.9
United Verde	1928	731,488	2.4 to 1	3.6	13.9
Copper Queen	1921	.....	2.8 to 1	4.7	8.0

Shovel Output

	Year	Type of Shovel	Dipper Capacity, Cu.Yd.	Cu.Yd. per Shovel-Shift	Per Cent of Cost: Labor	Per Cent of Cost: Supplies
Utah	1928	Electric	4.5	1,780	60.3	39.7
New Cornelia	1930	Steam	4.0	1,322	....	....
Chino	1929	Electric	4.0	1,190	....	....
Chino	1929	Steam	4.0	772	....	....
Nevada	1925	Steam	4.0	1,128	....	....
United Verde	1925	Steam	4.0	893	56.3	43.7
United Verde	1928	Electric	1.75	463	60.2	39.8
Copper Queen	1921	Steam	3.5	1,038	57.0	43.0

Table XII—Mesabi Range Open Pit Costs, 1928

	Cents per Ton
Labor	13.0
Supplies	13.2
Miscellaneous	14.7
Direct mining	40.9
Stripping	27.1
Total	68.0

ing, and also the amount of secondary drilling and blasting. Capacity of a given shovel will depend to a certain extent on the material being shoveled.

Topographic features have their influence on the method of laying out the transportation facilities. In part, waste disposal, length of tram, grade of tracks, and drainage are dependent on topography.

The shape of the orebody has an important bearing on mining methods and transportation. Of the other minor factors that determine costs, climatic conditions are, perhaps, the most important. Distribution of ore and waste within the orebody, and the number of classes of ore to be handled, have their influence on costs. These cost factors are outlined below:

1. Ratio of ore to stripping
2. Character of ground
   (a) Amount and method of drilling
   (b) Explosives
   (c) Secondary breaking
   (d) Speed of shoveling
3. Topographic features
   (a) Waste disposal
   (b) Length of tram
   (c) Drainage
4. Shape of orebody
5. Climatic conditions
6. Miscellaneous

In the past few years the most important reduction in costs of power shoveling has been effected by changing the railroad-type shovel to a caterpillar-mounted shovel. Electricity has played an important part in increasing efficiencies. Electric shovels are gradually replacing the older steam shovels in many power-shovel operations. Based on records from metal mining and nonmetal mining operations, the change has resulted in a saving of from 25 to 30 per cent of the direct operating costs. Electrification of the haulage and transportation facilties in and about the pit is proving equally effective in reducing direct transporation costs per ton. The costs presented have been summarized in Table XI.

Table XIII—Summary of Surface Mining Costs

Mine	Date	Cents per Ton: Direct Mining	Stripping	Total
Utah	1928	11.7	16.2	(a) 31.8
New Cornelia	1930	21.0	9.2	30.2
Chino	1929	31.4	45.0	76.4
Nevada	1928	21.4	40.0	61.4
United Verde	1928	30.2	150.0	180.2
Copper Queen (b)	1921	25.4	71.1	96.5
New Idria	1930	37.0	21.2	58.2
Mountain Copper	1931	....	....	38.6
Flowery	1928	37.0	16.0	53.0
Wasp No. 2	1911	....	....	53.5
Fresnillo quarry	1920	....	....	47.3
Fresnillo glory hole	1923	....	....	22.6
Mesabi Range	1928	40.9	27.1	68.0

(a) Overhead and fixed cost of 3.9c. per ton is included. (b) Estimated costs per ton from data on solid yards.

Reviewing these costs, certain general conclusions may be drawn. The first of these is that the mining cost by surface-mining methods results in a lower cost per ton than that shown by the general average of any of the six primary underground mining methods. Of the 350 important underground metal mines in the United States, only two show costs that compare favorably with surface methods of mining. These are Alaska Juneau, which produced about 40,000,000 tons at an average cost of 26c. per ton; and Miami Copper, which has produced in excess of 25,000,000 tons at an average of 40c. per ton.

The cost of producing ore by the surface glory-hole method and delivering it to the mill at Fresnillo is the lowest cost presented. On the basis of 6,000,000 tons of ore, the development cost was 3.7c. per ton. Apparently, no charge was made for stripping, which partly accounts for the fact that the cost is lower than comparative costs of Utah Copper and New Cornelia.

That the cost of power shoveling on the Mesabi range is almost identical to the average of the Chino and Nevada costs should be of interest. On the other hand, the Mesabi range cost is more than twice as much as the Utah cost.

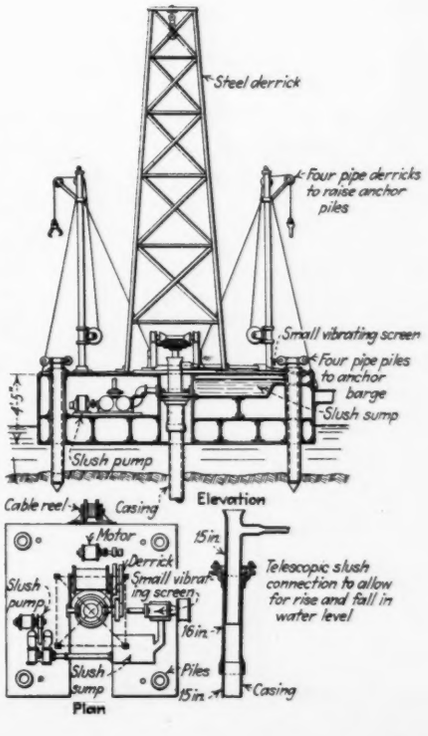
Production on an enormous scale is not entirely essential for low cost by surface-mining methods. The minor operations such as Mountain Copper, New Idria, Flowery, and the old Wasp No. 2 obtained costs that compare favorably with any of the large operations with the exception of the Utah Copper and New Cornelia. A large tonnage is advantageous in spreading overhead. Low costs on small tonnages are possible only where most of the other conditions are favorable.

Drill Barge Effects Economies

SUBMARINE DRILLING is difficult and expensive. Pile-driven drill locations and runways must be prepared to accommodate and keep the drilling machinery in perfect alignment and free from excessive vibration, and to facilitate handling of materials—one of the conventional but costly methods developed in recent years. The Jefferson Lake Oil Company, engaged in exploring by core drilling the sulphur-bearing strata below the bed of Lake Peigneur, near New Iberia, La., has developed a method that is destined to revolutionize submarine drilling in shallow water. Unlike the method mentioned, drilling is done from a drill barge, designed by Edward Laguerre, general superintendent, which at every change of location saves from $4,000 to $5,000, or the cost of preparing a new pile-driven drill location. A general arrangement of the barge is shown in the sketch presented herewith.

The all-welded steel hull houses the steel derrick; the drilling mechanism; and a slush sump, in closed circuit with a small sand pump, installed below the sump bottom to provide a moderate head on the suction, and a vibrating screen for grading the drill slush. Materials and tools are stored in the interior of the hull. Four heavy piles, made from pipe and equipped with eye-clamps, are used to anchor the drill barge. These are removed by small winch-operated derricks whenever a new location is desired. The most difficult problem—that of providing a watertight joint for circulating drill slush in the event of a sudden rise and fall of water level—was solved by the telescopic joint secured to the well casing.

Electric power from the end pole of the transmission line is transmitted through a submarine cable held by a reel bolted to the side of the hull. The rectangular slot in the hull permits the barge to be withdrawn after a well has been completed.



Mining and Furnacing Quicksilver Ore

Geology and Development of the First Regular Producer in the New Quicksilver Region in Arkansas

Noel H. Stearn

IN THE recently discovered Arkansas quicksilver district, the first prospect to be put into regular production is known as the Parnell Hill mine of the Southwestern Quicksilver Company. Situated in Section 6, Township 7 South, Range 25 West, near the junction of Cowhide Creek with the Little Missouri River in Pike County, Ark., it lies in about the middle of the west half of the narrow "cinnabar zone,"[1] 30 miles long, which extends across Pike County into Howard and Clark counties. The mine, 6 miles west of the main road from Murfreesboro to Hot Springs, is reached by a recondary road down Cowhide Creek Valley from a point 9 miles north of Murfreesboro, the nearest town. Glenwood, Ark., on a branch of the Missouri Pacific, 22 miles northeast of the mine, is the nearest railway.

[1]W. M. Weigel: *Eng. & Min. Jour.* Dec. 14, 1931, Noel H. Stearn: *Min. Jour.* (Ariz.) June 15, 1932, George C. Branner: Arkansas Geological Survey, I.C. 2.

In September, 1931, Leo Yount, who has had direct superintendence of the development, began prospecting on Parnell Hill by digging test pits and trenches on showings of cinnabar mineralization. His initial work soon disclosed perplexing local intricacies of structure that induced a detailed geological survey of the hill and its vicinity, and ultimately of the entire west half of the cinnabar zone. Much of the field mapping for this survey was capably done by L. L. Palmer, resident geologist for Southwestern Quicksilver, and Dr. J. M. Hansell.

A map of Parnell Hill and the adjacent area is shown in Fig. 1, on which map are recorded the topographic features and the observed details of surface geology. Obviously, the area lies in a region of submountainous topography characterized by ridges elongated in a direction slightly north of east. The total relief is about 330 ft. from the Little Missouri River to the top of Red Bluff. The local irregularities in topography, such as the patterns made by the stream courses and gullies, were soon recognized to have such definite geological significance that they were all carefully mapped.

Fig. 2 shows the present working interpretation of the observed facts, in terms of areal and structural geology. The area is underlain by a series of sandstone and shale formations. The sandstone formations consist of massive, medium to fine, even-grained, usually dark-gray beds, ranging in thickness up to 16 ft. Thickness of the beds is generally about 2 to 3 ft., with shale partings from a fraction of an inch to several inches across separating them. Being cemented with silica almost to a quartzite, the sandstone responds to weathering by shattering into angular blocks and fragments. This material forms talus on the steep slopes of the ridges, whose cores are always of massive sandstone.

The shale formations show a variety of characteristics, ranging from platy fissile yellow argillaceous shale, through tough blue-gray clay shale and hard black sandy shale, to massive shaly sandstone. Within the shale formations, sandstone lenses occur that range from fractions of an inch in thickness to massive series of beds measuring several feet across.

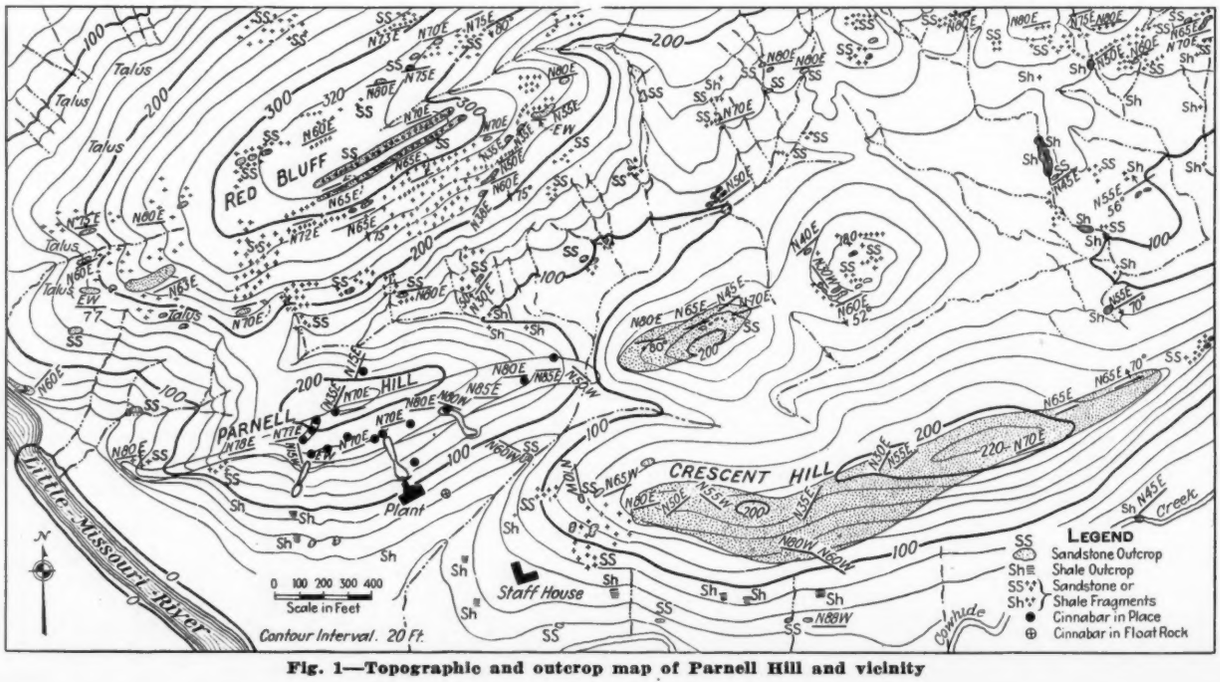


Fig. 1—Topographic and outcrop map of Parnell Hill and vicinity

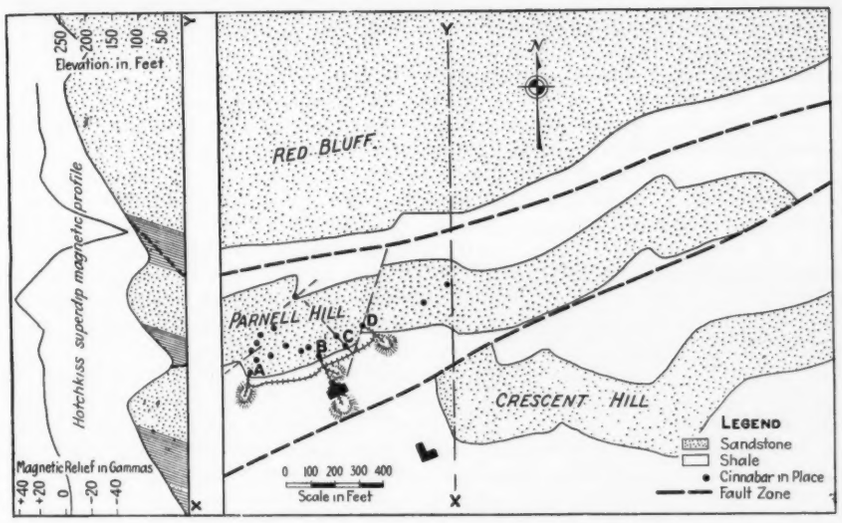


Fig. 2—Geological map and cross-section of Parnell Hill and vicinity

According to Miser and Purdue,[2] the sandstone in this area is part of the base of the Jackfork formation, and the shale part of the top of the Stanley formation. From evidence assembled by this survey, to assign both the sandstone and the shale on Parnell Hill to the same formation, either Jackfork or Stanley, seems more logical.

Regionally, the cinnabar zone lies on the Athens Plateau, which forms the south flank of the Ouachita Uplift, a large anticlinorium in central Arkansas having an original strike about N. 75 deg. E., but which had its backbone broken at a mature stage of its development, so that the east end now strikes about S. 65 deg. E. Because the cinnabar zone lies west of the break in the anticlinorium, its regional strike is N. 75 deg. E. The seeming intricacies of local structure become readily understandable when referred to the regional structural history.

The regional folding that formed the Ouachita Uplift has caused the formations in this particular area to stand at steep angles. The beds dip southward at an angle of about 75 deg. Various types of evidence, no separate item of which can be considered altogether unambiguous, combine to indicate that the tops of the beds are toward the south.

The breaking of the regional anticlinorium has caused an east-west shortening, which is reflected in this area by much local faulting and cross folding. The magnitude of this folding and faulting is shown in Fig. 2. Between Parnell Hill and Red Bluff, the presence of a major fault is hypothesized on the changes in the stratigraphic sequence. Although the evidence for this fault is not direct, magnetic profiles across the area, made with the Hotchkiss Superdip, give magnetometer indications tending to substantiate its presence. For the fault between Parnell Hill and Crescent Hill the evidence is obvious.

With the foregoing background Parnell Hill itself may be understood much more easily. Roughly cigar-shaped, measuring 750 ft. across and 1,800 ft. long, it rises 237 ft. above the high-water level of the Little Missouri River. It is cored with a massive sandstone formation about 330 ft. thick, which dips 72 deg. south, strikes N. 75 deg. E. with the long axis of the hill, and is flanked on the north and south by shale. Once presumably continuous with those of Crescent Hill, the formations of Parnell Hill appear to have been broken off and wedged between Crescent Hill and the huge mass of Red Bluff. The apparent east-west lateral displacement along the fault between Parnell Hill and Crescent Hill is about 1,750 ft., and the total apparent east-west shortening in the area, as indicated by the crumpled Parnell Hill-Crescent Hill sandstone horizon, is about 2,180 ft.

On the west end of Parnell Hill itself is a remarkable crossfold, the axial plane of which strikes N. 32 deg. E. and the axis of which is nearly vertical. The hill is riddled with fracture systems, along which local movements have ranged in magnitude from a few inches to tens of feet. Two sets of fractures seem to dominate, one striking about N. 30 deg. W. and dipping steeply eastward, and the other set striking about N. 30 deg. E. and dipping steeply westward.

The first showings of cinnabar to be discovered on Parnell Hill were on the south slope in float rock, which was traced to a massive shattered sandstone bed on the south flank of the hill. The trench dug on this bed was named the "Bloody Cut," from the startling crimson color it showed after rains. The cinnabar occurred only as fracture-filling material, and, at this point, seemed to saturate the fractures. This observation led to an exploration campaign designed to define and follow the shattered zones, which proved successful in uncovering much unsuspected ore. The discovery of the highly mineralized cross fold on the west end of the hill led to further exploration along its axis. Although no surface showings of cinnabar, even in float rock, had been found on the north slope of the hill, a test pit on the northeast extension of the axis of the cross fold disclosed cinnabar mineralization, and proved that the ore-bearing solutions had permeated the fractures across the hill.

After the fracture association of the cinnabar had been established, a concentration of cinnabar was considered possible at any point where a cinnabar-bearing fracture in the sandstone met the contact of the sandstone with the shale that flanked it on the south and under which it dipped. Accordingly, from a point far enough down the hill from the "Bloody Cut" to cross the shale-sandstone contact, a trench was cut northward into the hill, which disclosed at the contact a layer of almost pure cinnabar and a zone over 14 in. wide that was richly mineralized.

Control of the ore deposition seems to have been mainly structural. Four different types of structural control have thus far been encountered. All of them suggest that the ore deposition was definitely influenced by zones of shattering formed at the time of the east-west shortening in the area.

At *A* (see Fig. 2), the ore is found around the sharp unbroken nose of a cross fold, and in the shattered trough associated with it. The cross fold extends across the hill in a direction N. 32 deg. E., shattering the sandstone formation throughout a zone about 135 ft. wide and 630 ft. long. Cinnabar mineralization is found erratically throughout the entire zone.

At *B*, the ore is at the contact of the shale with the sandstone. The ore deposition is by no means constant along the strike or dip of this contact, and so some other factor must enter into the control of this particular deposit. This factor is thought to be found in the presence at *B* of an incipient drag fold in the shale.

At *C*, the ore is in the form of an irregular chimney in a zone shattered by a northwest fracture, along which displacement of 2 or 3 ft. has taken place. Here the cinnabar forms the matrix in a sandstone breccia, and the resultant product contains as much as 38 lb. mercury to the ton of run-of-mine ore.

At *D*, the ore occurs in the shattered fractures east of a fault striking N. 30 deg. E., along which several feet of displacement have taken place. Here the ore is leaner, hand picking yielding a product with about 15 lb. mercury per ton of ore.

The cinnabar on Parnell Hill is generally coarsely crystalline. Half-inch crystals have been found, and quarter-inch crystals are common. The crystals are frequently perfect double rhombohedral twins. Associated with the cin-

[2]H. D. Miser and A. H. Purdue: "Geology of the DeQueen and Caddo Gap Quadrangles, Arkansas," U.S.G.S. Bulletin 808.

nabar the following minerals are found:

1. Much dickite; a soft hot-water kaolin which feels like talc, usually powdery white but sometimes flaky yellow when stained with iron oxide.

2. Some vein quartz of the high-temperature variety.

3. Much iron oxide, with various degrees of hydration. The suspicion prevails that some of this represents the oxidation of iron sulphide that probably accompanied the cinnabar mineralization, and the presence of which is anticipated below the oxide zone.

4. One small showing of stibnite with stibiconite.

5. An arsenic mineral, such as realgar or orpiment, the presence of which is supected but has not been proved.

After exploration had disclosed sufficient showings of ore to justify the installation of a pilot plant, a plan of selective mining was formulated to permit (1) simultaneous development of four of the most promising ore showings; (2) separation of the ore into high and low grades; (3) immediate reduction of the high-grade ore at the pilot plant, and (4) convenient storage of the low-grade ore. The shape of the hill is especially adapted to this plan. About 100 ft. above the base of the hill and 115 ft. below the top, an 18-in.-gage track is laid along the hillside for 850 ft. This is high enough to permit the handling of the high-grade ore through the plant entirely by gravity. It is also high enough to permit gravity handling of the low-grade ore now being stored in dumps below the track, by another plant that can ultimately be built even lower on the hillside than the pilot plant. At the same time, the track is low enough on the hillside to be below the shale-sandstone contact, and to have potential ore reserves above it which, in process of mining, can be handled by gravity.

From the track level three cuts were made directly into the hill (at *A*, *B*, and *D*, in Fig. 2), crossing the shale-sandstone contact, and encountering the cinnabar mineralization on vertical faces of about 50 ft., which faces increase as the cuts are carried on into the hill. At *A* and *D*, mining consists simply of ordinary quarrying operations, supplemented by careful hand picking to separate the rich ore, which is trammed to the plant. The low-grade ore is trammed to storage dumps immediately in front of the cuts.

At *B*, the open cut was stopped when it encountered the sale-sandstone contact, and drifts were run east and west along the contact. Because these drifts are partly in shale, they are timbered with semi-square-sets, with lagging above and on the shale hanging-wall side. Inasmuch as the footwall consists of a massive quartzitic sandstone dipping 72 deg. S., no uprights are necessary on the footwall side, the cross members being fitted into notches blasted into the sandstone. From the west end of the drift a north crosscut reaches a point beneath the "Bloody Cut," from which the ore is removed through the drift.

At *C* blasting on an unmineralized fracture disclosed that it led to a brecciated zone richly mineralized. From the east drift, a crosscut designed to reach a point beneath this zone encountered a rich chimney-shaped orebody in a brecciated zone pitching at 80 deg. southeast along a minor fault. This high-grade orebody was stoped out to the surface. Then, from the level of the crosscut, an inclined winze was sunk by hand for 57 ft. on the orebody. The vertical continuity of the chimney to date is 107 ft., and the ore still persists in the bottom of the winze. This opening is all in the sandstone formation, which seems to stand satisfactorily, so that no timbering has as yet been found necessary.

Directly south of the cut at *B* in the middle of the south slope of the hill, the pilot plant was installed under the direction of C. Hyde Lewis, engineer for H. W. Gould & Company. It consists of a Gould Improved quicksilver furnace, the rotary kiln of which is 35 ft. long and 2 ft. in inside diameter, equipped with automatic ore feed and dust chamber, and a fuel-oil burner. The kiln rotates at 1¼ r.p.m. The plant is powered by a 25Y Fairbanks-Morse semi-diesel engine, which drives a 25-kw. generator, supplying power to a 10-hp. motor for the crusher, a 2-hp. motor for the ore feed, a 5-hp. motor for the compressor, a 1½-hp. motor for the fuel feed, a 3-hp. motor for the blower, and a 2-hp. motor for the kiln rotator, as well as lights and electric refrigeration for the plant and staff house. The condensing system consists of 200 ft. of 12-in. cast-iron pipe, arranged in four pairs of vertical stands with four drips to return the quicksilver, a redwood air chamber, and a 20-ft. redwood stack.

Because the cinnabar is not associated with any complicating minerals, extraction is simple. From the track level the high-grade ore direct from the mining faces is dumped into a 60-ton coarse-ore bin. Through a gate in the bottom of this bin the ore passes across a 1-in. grizzly, the coarse ore going through a jaw crusher set to crush to 1 in., and drops into a 45-ton crushed-ore bin. A short chute from this bin leads directly to the automatic feeder, and thence into the rotary kiln, where it encounters a temperature of about 1,600 deg. during the 35 min. required for passage. From the lower end of the kiln the burnt rock drops into a bin that can be emptied into a tram car running to a burnt-rock dump just below the plant.

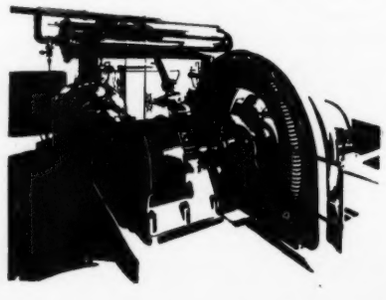
The water, mercury, and sulphur dioxide vapors from the kiln are boosted into the condensing system by a blower. They enter the cast-iron pipes at a temperature of about 400 deg.; so to insure complete quicksilver recovery, as quick and as complete a cooling as possible must be effected. The condensing system when installed was equipped with a water spray inside the second stand of the cast-iron condensing system. Only about 5 per cent of the recovered quicksilver was delivered by the first return, 80 per cent came from the second and third returns, 15 per cent from the fourth return and the tank, and the gas was about 25 deg. above summer atmospheric temperature when it issued from the stack. Almost all the quicksilver came out as flour mercury, which was so fine and so mixed with soot that extraction on the hoeing table was difficult, even with the use of much unslacked lime.

Ralph Cranston, plant superintendent, and Mr. Yount succeeded in overcoming these difficulties by eliminating the inside spray, and substituting an outside drip on the first five stands, which kept the outside surface of the iron pipes covered by a thin film of water. Also, an inside spray was installed in the base of the stack. The beneficial effects of these alterations on the cooling curve of the gases can be seen in the fact that the gases drop 90 per cent of their quicksilver in the first two returns, 5 per cent in the third, 4 per cent in the fourth, the remainder in the air chamber, and issue from the stack at a temperature only about 1 deg. above summer atmospheric temperature. The quicksilver is returned almost entirely in liquid form rather than as flour mercury, and much less unslacked lime and labor are required to clean it.

The tenor of the ore on which the plant was started amounted to about 12 to 15 lb. metallic mercury per ton of ore; but present runs yield about 38 lb. metallic mercury to the ton. The tenor of the low-grade ore on the storage dumps is erratic, and no adequate sampling has been done.

The pilot plant was put into operation on April 21, 1932. Its average throughput is 12 tons of ore per day, at a plant cost of $0.942 per ton. This figure does not include depreciation and overhead charges, but includes the following items:

	Per Ton	*Per Day*
Labor (crushing and furnacing)	$0.552	$6.63
Fuel (furnace and engine)	0.233	2.80
Lubricating oil	0.092	1.10
Water (pumping cost)	0.035	0.42
Lime	0.01	0.12
Miscellaneous	0.02	0.24
Total	$0.942	$11.31



Boltless Cylindrical-Mill Lining

E. D. McIntyre

Superintendent of Reduction Plant, New State Areas, Ltd., Johannesburg, Transvaal

THE wave type of tube-mill liner was designed to grip the load or grinding medium, thus preventing slip and insuring a vigorous inter-load action. A considerable amount of the work done in the tube mills on the Witwatersrand is by causing the load to cataract. This cataracting is governed by the peripheral speed of the mill and the angle of liner grip, which effects fall of the grinding or crushing medium on the ore. Given sufficient peripheral speed, a liner with too positive a grip will throw its load onto the liner on the opposite side, above the pulp level, thereby wasting power that should be consumed in reducing ore. A liner that is too flat will drop grinding medium on grinding medium, or give a cascade effect only under ideal conditions. The pulp at the base of the load should be of sufficient volume so that energy imparted by the cataracting grinding medium will be absorbed in it, although it should not be of such a volume that would cause energy to be "smothered." The aim, therefore, is to provide a type of liner that will give a maximum spin to the ball or pebble, and insure the correct drop when the liner is half worn.

Fig. 1—Portion of liner with jacks in position. Mill ready to be turned over. Liner blocks to complete the six circles are being placed in mill.

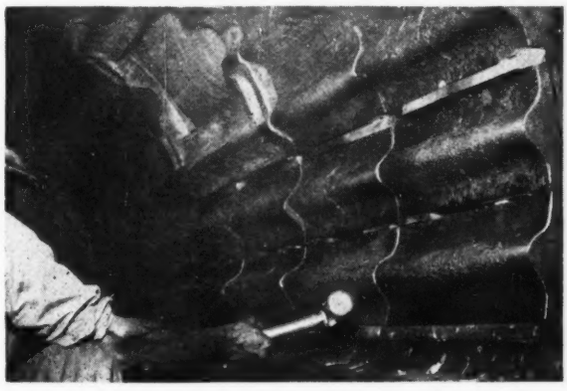


Fig. 2—Keys being temporarily tapped up prior to removal of jacks. Spacing bars visible two joints below key, and small wedges in joint below.

The corrugated liner to be described, a common type, is mostly made of hard white iron with sufficient chrome, or manganese, or both, to toughen it. Each block weighs from 120 to 175 lb. Dimensions vary according to requirements, the average being about 19 in. long by 10½ in. wide by 4 in. deep, rounded off to 1½ in. at the fitting edges. The weight of a complete set of liners for a 6 ft. 6 in. by 20 ft. mill is about 17 tons.

In this liner, the corrugations are regular, except that the concavities are generally narrower than the convexities, to prevent undue wear at the joints, which wear is influenced by the size of the grinding medium. The ends of the blocks are rough, and the joints abut. The blocks from the inlet end of the mill are thicker than those from the center outwards, so that the liner may be completely worn out before being discarded. Greater wear takes place at the inlet end, so the first half of the liner may be ¾ in. thicker there. With good operation it should retain a corrugation until ready for discard, but it will be considerably flattened, measuring ¾ in. at the edges to 2 in. at the highest point, though this shape will not obtain with poor-quality castings.

Fig. 3—After removal of the jacks the keys are being finally driven home. Note tilt given to block below the wedge, to make the final space wedge shaped, to hold key.

Liner blocks are put in a sliding tilting cradle at the foundry, which passes them to and fro before an emery wheel. This straightens the edges, and splays them so that the radial lines are correct. A gage is passed over them before acceptance.

The door of the tube mill is the starting point for lining. Frame pieces, cast to the pattern adopted, fit around the door opening, allowing plenty of clearance. Assuming that the door measures 24x18-in., each half-frame would be two block lengths of 19x1½-in. units, or 15 in. with the door portion housed out. These frames put together show 6 to 7 in. of metal around the door opening. A door piece to complete the liner is bolted to the door cover, and this piece must have the sides and ends well splayed, to facilitate withdrawal for inspection purposes.

Jacks are necessary for lining. These are usually made of rock-drill screws fitted into pipes, having a hollow foot and head to go over the rounded liner. A length of 3-in. pipe, with a pronged footpiece welded in at one end and the nut for the rock-drill screw (2 in.) welded in the other, makes a good light jack; but both nut and footpiece must be collared over the pipe to take the thrust. The screw must have a plain, enlarged square, near one end, through which holes are bored for Tommy-bar operation, or it may be turned with a spanner or wrench. Above the square the screw is turned plain for about 2 in. This fits easily into a shallow U-piece, corresponding to the footpiece external to the pipe. Six of these jacks are necessary when lining a tube mill, or one to each row of blocks put in at one time.

Lining is started with the door at the

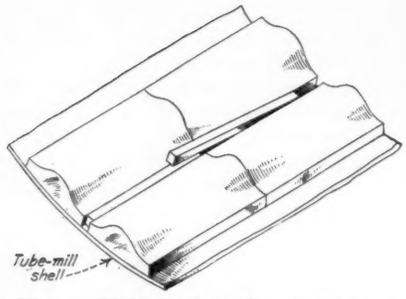


Fig. 5—Method of keying wave-type liner blocks

bottom. The frame pieces are put carefully in position around the door, making sure that all are straight. Six circles of blocks may be put in thus, laying them on both sides of the door piece, up to the center of the mill, when a jack is spanned across, gripping the top blocks of each row. As the jacks are tightened, the blocks must be hammered together, so that all fit tight. Sufficient are put into the tube at each end, and clear of the jacks, to complete the six circles when the mill is turned over. After making sure that all jacks are secure, the mill is turned around with chain tackle. As it revolves, blocks are laid to complete the circles; and then, when the last is in, a space of about 1 in. should be available for keying purposes. The keys are inserted in the right-hand corner at an angle, downward. This allows the block completing the next circle to pass under it. As each row is completed, keys are inserted, and temporarily tapped up. When the six rows are laid, all keys are slogged home, and the jacks removed, when the next six rows may be started. During the keying operation the jacks are slackened just enough to allow the tightening movement to take place over the whole circle.

Tapered bar iron is procurable 1½-in. wide and in varying thicknesses, which compensate for inequalities in keying spaces. Sizes of tapered iron start from 0 to ½ in., rising by eighths of an inch to 1 in. If ½-in. or more keying space is left, the liner will not drop because of weak keys, but the edges must be well fitted. Should the keying space be too wide, packing pieces may be used in the final few joints, to bring the last ones closer. Some operators insert thin shims between the left-hand joints of the last block or two, whereby the opening of the last joint will be wider on the right, or keying, hand, thus insuring a driving fit to the key.

Fig. 4—Final circles being keyed up. Those to the left are finished; those to the right are being keyed. Outlet screen and deflecting blocks in distance.

The time taken in lining a 6 ft. 6 in. by 20 ft. tube mill is seven hours; and the same mill can be stripped of old liners, relined, and the load put back for starting in eighteen hours, with a gang of ten kafirs.

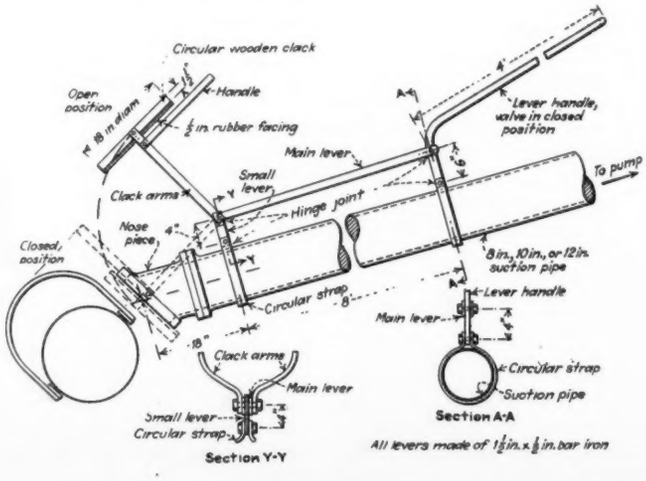
In breaking down an old liner, the blocks should be thin enough to permit them to be broken with sledge hammer. A weak section is selected and struck with the sledge hammer, standing well to one side. Fracture of the block should allow that row to fall, but some blocks may have to be pried down with a pinch bar, if pieces of metal become wedged in the ends. Should the liners be unbreakable, or if the liner has to be removed before it is worn out, the keys must be drifted out, using tools the shape of which will be suggested to the operator.

■ ■ ■

Hand-Operated Clack Valve For Pump Priming

A CLACK VALVE, easily operated by hand and particularly suitable for priming a pump attached to a movable suction pipe, is shown in the accompanying drawing, which, with the following details, was supplied by W. E. Sinclair. The valve simplifies the process of priming and starting up gravel pumps used to elevate "wash" on the alluvial or placer mines in Nigeria.

In employing the device, the pump suction pipe is partly lowered into the water in the sump, after closing the valve. While the priming of the pump proceeds, the valve clack is held tight against the suction pipe nosepiece by downward pressure on the lever handle. The lever handle is sometimes held in this position by means of a hook contrivance (not shown in the drawing), instead of by hand. Priming is performed in the usual manner, and, when completed and the pump started, the valve is opened by releasing the lever handle. Afterward, the handle is drawn down, and the clack pulled up, clear of the suction opening, and held in the open position by the lever handle pressed down to its extremity. The suction pipe is then lowered down into the sump to any depth that may be required.



Copper Matte Composition—II

C. K. Avetisian

THE FIRST EXPERIMENTS with Kalata matte proved the oxygen content to be overestimated, as it also was in Mostovich's tests based on the associated iron. The results obtained are given in Table II.

Samples were taken with a spoon. Whatever changes may have occurred during cooling—such as precipitation of metallic copper or formation of compounds high in sulphur—presence of Cu_2S, FeS, ZnS, and ferric oxide might be assumed in the liquid phase. Calculating the sulphur contained in these compounds, stable at high temperature, the amount of iron not combined with sulphur may be computed. For matte No. 1 this figure is 9.83 per cent Fe, combined with 4.86 per cent oxygen; for matte No. 3 the figures are 8.64 and 6.99 respectively. This gives for matte No. 1 a ferric oxide with an oxygen content of 33.1 per cent, and for matte No. 3 an oxide with 44.7 per cent oxygen; whereas the higher oxide Fe_2O_3 contains only 30.06 per cent oxygen. This raises the question if determination of oxygen by the Hampe method is correct.

Table II—Analysis of Matte Samples

	Matte No. 1	Matte No. 3
Copper, per cent	23.27	24.65
Iron	41.83	40.04
Zinc	2.62	2.35
Sulphur	25.46	25.34
Oxygen	4.86	6.99
Silica	0.68	Not determined
Lime	0.51	
Total	99.23	99.37

The assumption is made that nothing but oxygen and sulphur are lost in the matte during the reduction. Does not the matte contain other volatile compounds? The presence of zinc in the two samples suggests that it does. Zinc in the matte occurs as sulphide, which is not volatile below 1,000 deg. C.[25] The vapor pressure of metallic zinc, at 830 deg. C, may be determined by the formula $\log P = 6365 : T + 8.17$, where T is the absolute temperature.[26] At $T = 273 + 830 = 1{,}103$ deg. abs. we have $\log P = 2.4$, and $P = 250$ mm. mercury, indicating that volatilization of metallic zinc is possible at this temperature. Formation of metallic zinc may take place in two ways: by reduction of the sulphide by hydrogen, as proved by Morse, and by reaction with metallic iron: $ZnS + Fe = FeS + Zn$.

When matte is reduced with hydrogen, an amount of metallic iron corresponding to about 8 to 10 per cent of the weight of the matte is produced, or 3 to 4 times the amount of zinc. Johnson has shown that zinc sulphide is completely reduced by metallic iron at 1,200-1,250 deg. C. This reaction may also, to some extent, take place at 830 deg. C. Determination of zinc in the residues showed a loss 0.9 per cent, or 34.4 per cent of the total zinc content of matte No. 1, 0.59 per cent and 25.1 per cent in matte No. 3. Deducting these losses from the oxygen content, the corrected figure for matte No. 1 is 4.86 — 0.90 = 3.96 per cent oxygen, and for matte No. 3 6.96 — 0.59 = 6.37 per cent. This gives an iron oxide with 28.7 per cent oxygen in matte No. 10. In matte No. 3 the corrected figure is still high—42.5 per cent oxygen in the iron oxide. Further investigations showed a decrease in the amount of zinc after reduction with hydrogen, in the original matte as well as in the residue from the cyanide leach. Table III gives the figures with correction for zinc losses. Examining Mostovich's data on oxygen in matte,[8] I found an excess of oxygen in samples containing from 0.6 to 3.45 per cent zinc. This shows that the Hampe method cannot be used without modification.

Determination of oxygen in matte No. 3 as well as in all other mattes has shown the oxygen content to be very high, even when correcting for volatilized zinc. Moreover, I did not always succeed in balancing oxygen before and after leaching the matte in cyanide, which was possible with all other elements. This points to the presence of other volatile elements, such as arsenic and antimony.

In considering the composition of the iron and copper compounds dissolved in KCN, interesting data may be obtained by computing the percentage of all dissolved elements, assuming their total to be 100, as shown in Table IV.

This table shows that the compound dissolved in KCN is the same for all mattes with copper content below 62.8 per cent. Its analysis varies from 60.7 to 61.7 per cent copper, 14.7 to 15.7 iron, and 23.6 to 24.0 per cent sulphur.

Table III—Results Leaching 1-Gram Samples in 7 Per Cent KCN

	Matte (Pct.)	Matte (Grams)	Grams Dissolved	Grams in Residue	Total Accounted For
Matte No. 1					
Cu	23.27	0.2327	0.2383	None	0.2383
Fe	41.83	0.4183	0.0603	0.3670	0.4273
Zn	2.62	0.0262	Trace	0.0262	0.0262
S	25.46	0.2546	0.0945	0.1775	0.2720
O_2	3.96	0.0396		0.0468	0.0468
SiO_2	0.68	0.0068		0.0050	0.0050
CaO	0.51	0.0051		0.0050	0.0050
Total	98.33	0.9833	0.3931	0.6275	1.0206
Weight of residue: 0.6186 grams.					
Matte No. 2					
Cu	23.85	0.2385	0.2242	0.0175	0.2417
Fe	41.13	0.4113	0.0532	0.3460	0.3992
Zn	2.67	0.0267	Trace	0.0267(a)	0.0267
S	25.84	0.2584	0.0858	0.1750	0.2608
O_2	5.22(b)	0.0522(b)		0.0566	0.0566
SiO_2	0.68	0.0068		Not determined	
CaO	0.38	0.0038			
Total	99.77	0.9977	0.3632	0.6218	0.9850
Weight of residue: 0.6176 grams.					
Matte No. 3					
Cu	24.65	0.2465	0.2460	Trace	0.2460
Fe	40.04	0.4004	0.0636	0.3375	0.4011
Zn	2.35	0.0235		0.0235(a)	0.0235
S	25.34	0.2534	0.0954	0.1580	0.2534
O_2	6.40	0.0640		0.0461	0.0461
Total	98.78	0.9878	0.4050	0.5651	0.9701
Weight of residue: 0.5752 grams.					
Matte No. 4					
Cu	55.20	0.5520	0.5235	0.0324	0.5559
Fe	19.23	0.1923	0.1311	0.0612(a)	0.1923
S	24.72	0.2472	0.2063	0.0409(a)	0.2472
Zn	None				
O_2	0.40	0.0040		0.0040(c)	0.0040
Total	99.55	0.9955	0.8609	0.1385	0.9994
Matte No. 5 (d)					
Cu	62.84	0.6284	0.6173	0.0136	0.6309
Fe	15.48	0.1548	0.1269	0.0279(a)	0.1548
S	22.39	0.2239	0.2170	0.0069(a)	0.2239
O_2	0.79	0.0079		0.0079(c)	0.0079
Total	101.50	1.0150	0.9612	0.0563	1.0175
Matte No. 6 (d)					
Cu	70.17	0.7017	0.7017	Almost completely dissolved	
Fe	4.30	0.0430	0.0113		
S	21.57	0.2157			
O_2	0.12	0.0012			
Total	96.16	0.9336	0.9336		

(a) By difference. (b) Not corrected for zinc content. (c) Not determined. (d) Contains no zinc.

In mattes with more than 62 per cent copper and with less than 15 per cent iron, this composition is not possible. Such a matte is almost completely dissolved in KCN. Below 62 per cent copper the amount of residue increases with decreasing copper content. In low-grade matte copper, iron, and sulphur therefore form definite compounds. In high-grade matte these compounds exist with an excess of Cu_2S, soluble in KCN; in low-grade matte with an excess of FeS and Fe_xO_y, insoluble in KCN. A similar deduction was made

Table IV—Percentages of Dissolved Elements

Matte No.	Per Cent Copper	Percentage Dissolved in KCN		
		Copper	Iron	Sulphur
1	23.27	60.7	15.3	24.0
2	23.85	61.7	14.7	23.6
3	24.65	60.7	15.7	23.6
4	55.20	60.8	15.2	24.0
5	62.84	64.2	13.2	22.6
6	70.17	75.2	1.2	23.6

by Gibb and Philp, who finally selected the compound $5Cu_2S.FeS$, with 71.9 per cent copper, 6.32 iron, and 21.78 per cent sulphur[6]. If my results be accepted, however, this compound must be rejected. The compound suggested by Borneman and examined by Reuleaux, $2Cu_2S.FeS$, with 62.66 per cent copper, 13.73 iron, and 23.72 per cent sulphur, most closely approximates the compound dissolved in my tests. To determine the composition of the soluble compound in Mostovich's tests his data have been compiled in Table V, with my calculation of its composition. On account of the loss of sulphur as sulphocyanide, the percentage of sulphur in the soluble compound is slightly lower than in my results. Nevertheless, the table gives full proof of my deduction regarding the presence of a definite compound with an approximate composition $(Cu_2S)_2FeS$, soluble in KCN.

When Yushkevich pointed to this compound, formed by heating Cu_2S with FeS, he did not have sufficient data on hand; the action of KCN solution had not been adequately studied, and this had led Mostovich to deny the presence of a compound $(Cu_2S)_2FeS$, although his own experiments confirmed its existence. The main reason for this denial was the variation in the ratio of sulphur combined with copper to that combined with iron, which, according to Yushkevich, was about 2:1. Thus the loss of a small part of sulphur as sulphocyanide proved a stumbling block for both investigators.

An interesting fact is that Gibb and Philp, studying the reaction between metallic copper and ferrous sulphide, found a matte of the following composition: 60.6 per cent copper, 17.8 iron, and 21.6 per cent sulphur. Baïkow and Trutnev obtained a matte with 58.87 per cent copper, 17.07 iron, and 20.99 per cent sulphur. Both of these approximate the given compound. The reaction of metals and sulphides may best be

Table V—Data of Mostovich's Tests

Smelter	Per Cent Copper in Matte	Mg. Dissolved in KCN Iron 1-Gram Sample			Composition of Soluble Material in per Cent		
		Copper	Iron	Sulphur	Copper	Iron	Sulphur
Pyshma	38.50	337.6	89.5	115.2	62.2	16.5	21.3
Bogoslow	38.57	347.6	113.1	117.1	60.1	19.6	20.3
Kalata	14.55	134.8	38.6	44.3	61.9	17.7	20.4
Ulejn	41.05	379.6	105.0	120.5	62.7	17.4	19.9
Dsansul	37.76	282.0	78.4	99.5	61.8	17.2	21.0
Tanalyk	12.25	88.8	30.8	30.07	59.3	20.6	20.1

expressed by the equations proposed by Yushkevich; $3\ Cu_2S + Fe = 2Cu_2S.FeS + 2Cu$ and $4Cu + 3FeS = (Cu_2S)_2FeS + 2Fe$.

Conclusions

1. The only stable compound formed by the reaction between cuprous sulphide and ferrous sulphide is $(Cu_2S)_2FeS$, soluble in KCN.

2. All copper mattes contain this compound. If the copper content is below 62.7 per cent, the matte contains an excess of ferrous sulphide and ferrite (magnetite), insoluble in KCN. Matte containing more than 62.7 per cent copper has an excess of cuprous sulphide, soluble in KCN. The amount of insoluble residue thus increases with decreasing copper content. Mattes with 62.7 per cent copper or more are almost completely soluble.

3. Cooling of matte, depending upon the rate at which it proceeds, causes precipitation of metallic copper, soluble in KCN; at the same time insoluble sulphides are formed, higher in sulphur than FeS, among which is $CuFeS_2$. This conclusion is based on the study of samples in which copper did not completely dissolve in KCN. The amount of metallic copper and $CuFeS_2$ in the samples examined was not sufficient to prevent identification of the compound $(Cu_2S)_2FeS$.

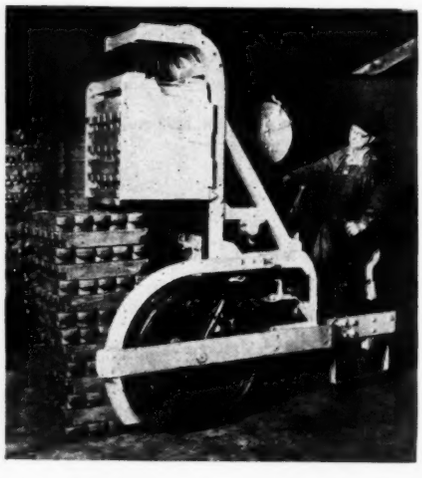
4. When matte and sulphides are dissolved in KCN, sulphur dissolves as sulphide and as sulphocyanide.

5. Hampe's method of determining oxygen, by reduction with hydrogen, gives too high results on account of reducible and volatile elements difficult to identify, such as zinc, antimony and arsenic.

Bibliography

1. *Metallurgie,* Vol. 3, p. 479 (1906).
2. Transactions A.I.M.E. Vol. 38, p. 424 (1908).
3. Transactions Petersburg Polytechnic Institute, Vol. 10, I, p. 309 (1908).
4. *Metallurgie,* Vol. 6, p. 619 (1909).
5. *Engineering and Mining Journal-Press,* Vol. 115, p. 1055 (1923).
6. *Metal und Erz,* Vol. 5, p. 99 (1927).
7. Transactions A.I.M.E., Vol. 39, p. 584 (1908).
8. Transactions Siberian Institute of Technology, Vol. 1, (1925).
9. Journal of Russian Metallurgical Society, Vol. 1, (1925).
10. Transactions A.I.M.E., Vol. 36, p. 665 (1906).
11. *Metal Industry* (London) Vol. 36, No. 19, p. 504, and No. 21, p. 553 (1930).
12. *Engineering and Mining World,* Vol. 2, No. 5, p. 270, (1931).
13. Journal of Russian Metallurgical Society, No. 5, (1911).
14. W. A. Wanjukoff: To the question of the affinity of elements at high temperature (Russian), p. 87, (1916).
15. E. Prost: "Metallurgie des métaux autres que le fer."
16. H. O. Hofman: "Metallurgie of Zinc and Cadmium," 1922, p. 11.

Efficient Pig-Lead Stacker



THE pig-lead stacker shown in the accompanying illustration was developed from the Collinsville-Newman pig-lead truck, at the smelter of the Bunker Hill & Sullivan Company, Kellogg, Idaho, writes A. F. Beasley, superintendent. In constructing the stacker, the wheelbase of the truck was lengthened and a hoist added.

Three loads of 35 pigs each, or a total of 105 pigs, may be placed in a safe, neat pile with the stacker, as shown in the illustration. A pile may be taken down as readily as it is formed. The stacker effects a decided economic saving, compared with manual stacking. Moreover, as the device operates on any relatively smooth floor, it may be used over a wide area, whereas a crane employed for the same purpose can serve only a limited area.

LEVEL DEVELOPMENT

Lucien Eaton

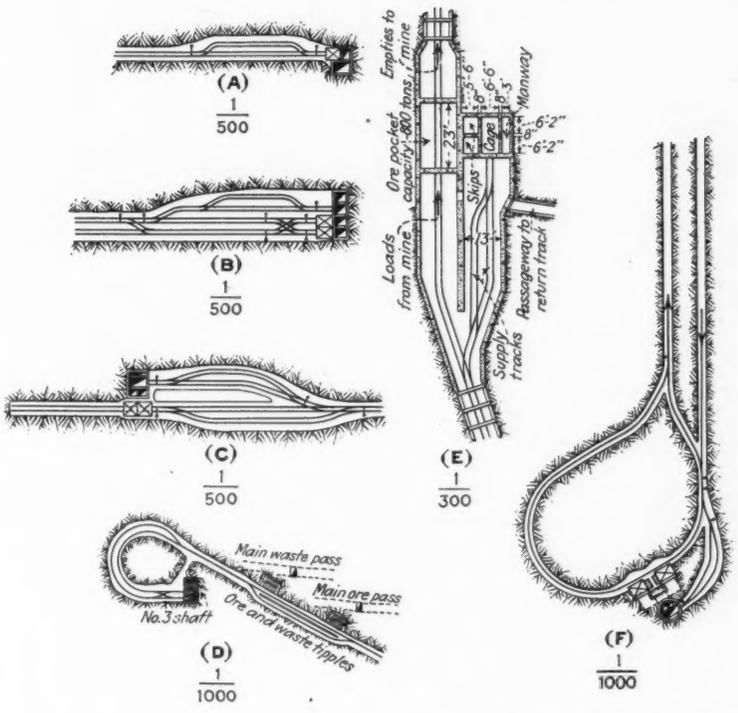


Fig. 53—Shaft stations

IN LAYING out underground development, the depth at which the first level will be turned off in the shaft, and the interval between levels, are determined by the size, shape, and position of the orebody and the mining system to be used. The first level is usually opened at such an elevation that sufficient ore is above it to justify the development. At the same time the maximum height of ore should not be too great for economical mining. Sufficient information is rarely available to determine either of these points, and the elevation chosen is largely a matter of judgment, or at best a compromise.

The number of levels that should be opened at one time depends on the rate of mining and on the time required to open a level and to prepare the ore above it for mining. Development should be kept far enough ahead to provide a sufficient number of working places at all times, and to give the new level an opportunity to drain thoroughly before mining starts, but not so far ahead as to tie up too much capital or to allow the timber to rot, except in those operations in which such development is necessary to prove ore reserves in advance.

To keep down the cost per ton for development and level equipment, the interval between levels is made as large as is compatible with convenience, safety, and economy in mining. A point is reached, however, where the saving per ton in the depreciation of the first cost of the level and its equipment is more than offset by the increase in the cost of intermediate development and of mining at long distances above the level, by the cost of tramming and transferring ore on intermediate levels, and by the maintenance charges on drifts, raises, and chutes. The more expensive the main-level development, the greater the level interval that is justified. In general, the economic limit is between 100 and 200 ft. However, notable exceptions may be cited, such as the diamond mines at Kimberley, South Africa, where the level interval is 600 ft. The shaft is a long distance from the orebody, and main-level development is elaborate. The ore passes are vertical, of large diameter, and unlined, and the ore is not abrasive. Under these conditions, maintenance is low. Another exception is the Miami Copper Company's mine at Miami, Ariz., where the ore is mined by undercutting and caving, and the most economical level interval is 300 ft. (Tech. Paper 314. A.I.M.E., p. 42, by F. W. Maclennan.) Others might be cited.

If the orebodies are small and scattered, the tonnage developed on a level may not be large, even though a large amount of drifting is required, and it is a temptation to increase the level interval. This entails risk, because the exploratory value of the drifting is considerable, and for the reason that every increase in level interval not only makes more difficult the finding of the downward continuation of the orebodies developed on the level above, but also increases the chance of missing some orebodies altogether. These objections may be overcome by driving intermediate levels in the ore formation, and transferring the ore to main-haulage levels, which are some distance apart. The cost of transferring the ore must be balanced against the cost of a completed level to determine which plan should be used.

If raises have to be timbered, wear on the timber increases rapidly when chutes are more than 100 ft. high, and becomes a serious matter if a height of 150 ft. is exceeded. This places a natural limit on the level interval for such methods of mining as top-slicing, sublevel caving, square-set, and horizontal cut-and-fill; and a similar limit is usually imposed on shrinkage stoping, but for different reasons.

The shaft station, or "plat," in its simplest form is merely a drift or crosscut leading away from the shaft; but more and more elaborate designs have been developed as the demand for better hoisting and transportation facilities has increased.

Shaft stations may be divided into two classes: (1) Those in which traffic to and from the shaft passes over the same route, which might be termed the intermittent, or single-track, type; and (2) those in which traffic to the shaft passes over one route and that away from the shaft goes by another. This might be termed the continuous or double-track type.

Most shaft stations are of the first class, because it is less expensive, and because its capacity is usually sufficient for the tonnage to be handled on any one level.

Examples of shaft stations are shown in Fig. 53. In Fig. 53a is shown a station designed for end-dump cars and a single skip. As the pocket holds only one skip load, dumping is intermittent, and the tramming crew does not wait for the skip, but leaves the loaded cars on the straight track and returns inside immediately with empty cars, taken from the siding. When several loads have accumulated, skip riders stop the skip at that level, dump the cars, and set out the empties on the siding. The switches are automatic. This design is suited to mines where a small tonnage

This is one of a series of articles on mine equipment.

is hoisted on each of several levels, and can be used with a cage as well as a skip.

In Fig. 53b is shown a somewhat similar station designed for hoisting ore with two skips and for handling supplies with a special cage. Tramming may be intermittent or continuous, depending on whether the service track is single or double.

A variation is shown in Fig. 53c, with two pockets, each holding one skip load, and with which gable-bottomed cars are used. A train of loaded cars is pushed across the pockets, and is left on the straight track beyond the shaft. The locomotive is uncoupled, picks up a train of empties on the siding, and returns inside immediately. The loaded cars are spotted over the pockets by a small air hoist, and are dumped one at a time, and the empty train is set out on the siding. One skip tender is employed. If the amount of ore trammed on the level is so small that the train crew has time to do its own dumping, or if a storage pocket is provided, the siding and the extra train of cars are unnecessary. Such a station, having storage pockets for both ore and waste, is shown in Fig. 53d. In both Fig. 53c and Fig. 53d the arrangement of tracks is complicated by the necessity of handling timber and other supplies with a cage on the same level.

In all of these stations of the first class the capacity is limited by the length of time required for tramming the train to the shaft, dumping the cars or switching, and returning to the nearest passing point.

In Fig. 53e and Fig. 53f are shown two stations of the second class, one with one storage pocket and the other with two storage pockets, in which capacity is limited only by the speed of dumping. Traffic is in one direction only, and can be almost continuous.

If the cars are dumped directly into a storage pocket, and if chunks of ore are present of a size large enough to cause trouble at the chutes below, the custom is to place a grizzly over the pocket. If the cars are dumped singly, the grizzly may be horizontal; but if a whole trainload is dumped quickly, one car after another, the grizzly bars should slope away from the track at an angle of 15 to 30 deg., so that the coarse pieces from the first cars will slide or roll out of the way and not be buried under the contents of the later cars. The chunks are broken in the interval of time between trains.

An example of grizzly construction for a storage pocket is shown in Fig. 54. The grizzly bars are I-beams covered by manganese-steel wearing plates. This construction is fairly expensive, but is justified if the ore is abrasive and the transportation cost of materials is high.

In shaft stations the openings usually have wide spans, which will stand unsupported only in very strong ground, but excessive widths can often be avoided by careful designing. Sometimes artificial support can be avoided through proper arching of the back, or in ground that weathers easily by Guniting; and, if it can be safely done, it is a great economy, but no unnecessary risks should be run in doing it. If support is necessary, the plat is usually timbered. Timber has the advantage of cheapness and convenience, but it is subject to decay, and danger of fire is always present. Chemical treatment may reduce these disadvantages, but the present tendency at stations where a large tonnage is to be handled is toward the use of concrete or steel or a combination of the two.

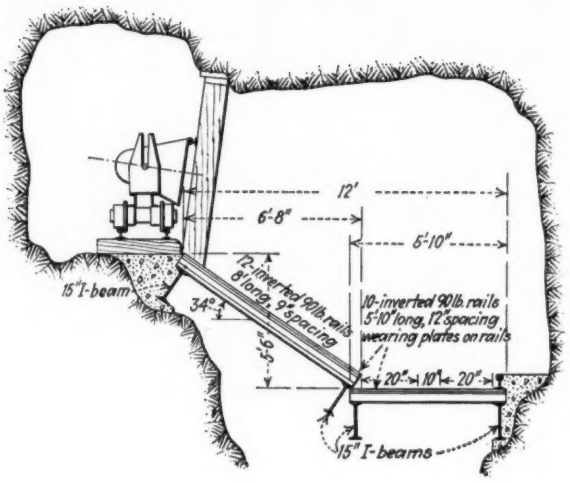


Fig. 54—Standard ore-pocket grizzly

Concrete may be continuous or intermittent, and in the last-mentioned contingency the ground between arches is Gunited or is supported by lagging of some kind. Reinforcing may or may not be used. When the concrete is poured in place behind forms, even if it is placed with compressed air, difficulty is experienced in completely filling the top of the arch. For this reason, arches made of pre-cast concrete blocks, or concrete walls supporting steel beams, are in favor. The walls may be made of monolithic concrete, or may be built of pre-cast concrete blocks. The latter type of construction is to be preferred if enough of it is to be done to warrant the equipment for making the blocks, because that practice avoids the need for forms, and because the concrete can be seasoned and tested before it is sent underground. Spaces behind the wall can be filled with concrete or with loose rock.

The back may be supported by arches made of concrete blocks or pressed-steel flanged plates, behind which loose rock or concrete is placed, or by steel beams, lagged with wood or with reinforced concrete slabs. Examples of different types of construction are shown in Figs. 55 and 56. In Fig. 56b is shown a patented type of arch made of pre-cast concrete blocks keyed together and tied with wire rope, which has gained favor in the Pennsylvania anthracite region.

Even in fairly strong ground a certain amount of weight on the supports develops, and some cushioning effect is often desirable. For this reason wooden

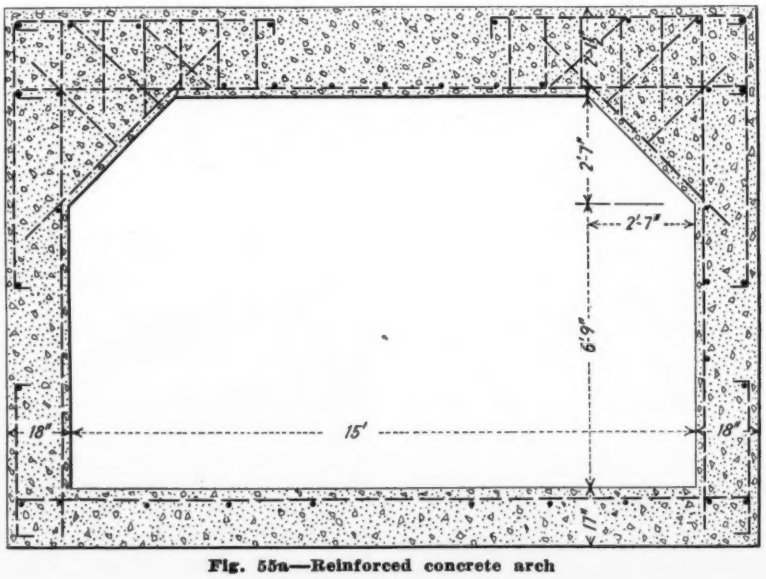


Fig. 55a—Reinforced concrete arch

blocks are inserted in the concrete arch, as shown in Fig. 55b, and a square timber may be laid along the top of the concrete wall under the steel caps, as shown in Fig. 56a. Where tremendous pressures develop at great depths, even in strong ground, as on the lower levels of the Calumet & Hecla mine, in northern Michigan, dry walls, laid up with layers of timber to tie them together and to provide cushioning effect, have been found a most effective method of ground support. (U. S. Bureau of Mines, I. C. No. 6526, 1931, by Harry Vivian.)

The use of steel supports underground has made greater progress in Europe than in the United States, on account of the higher cost of timber

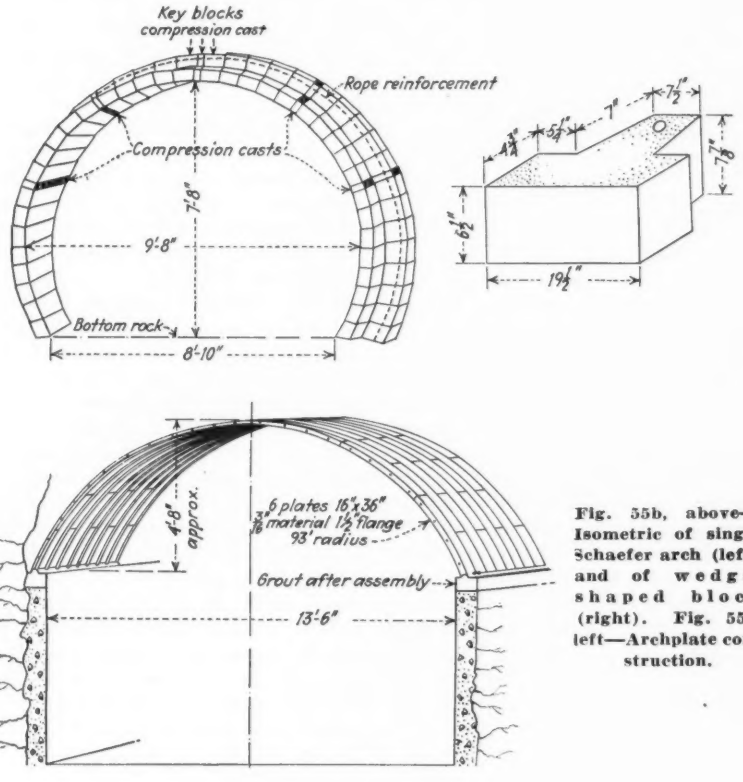


Fig. 55b, above—Isometric of single Schaefer arch (left) and of wedge-shaped block (right). Fig. 55c, left—Archplate construction.

there. Examples of steel sets are shown in Fig. 56b and Fig. 56e, in which the sets are rigid, and in Fig. 56d, where a cushion is provided under each leg and at the keystone of the arch. Steel sets may be lagged with plank, poles, light-steel rails, or reinforced concrete slabs. This type of construction, usually less expensive than concrete, gives excellent results, but the special shapes required are not always readily available.

In flat beds and in narrow steep veins, drifts are usually driven in the ore; but in wide veins and in long, massive deposits the main drifts are best kept in the footwall, with crosscuts from them across the orebody at intervals. By this procedure the main-haulage drifts can be kept reasonably straight, and timbering can often be obviated. Moreover, most of the chutes can be put in the crosscuts, and car loading on the main line can be avoided.

If the orebody is short and wide, somewhat the same result may be gained with less work in rock by driving the main crosscut across the middle of the orebody, and by turning off parallel drifts right and left from this crosscut at suitable intervals. These drifts to right and left should not be directly opposite each other, because this makes a poor track arrangement, and because the junction is a hard place to support, but they should be staggered enough to give proper clearance at the switches. If the radius of the curve at the turnoff is 80 ft., this offset should be 25 ft.; and if the radius is 45 ft. it should be 20 ft.

If the shaft station is of the continuous or double-track type, main-line traffic should preferably move in one direction only; and if this arrangement is feasible, the drifts and crosscuts constituting the circuit should be connected so as to form a loop or loops, but in most operations of the type here under discussion the branch drifts and crosscuts are preferably left with dead ends. The reasons for this are that curves and turnoffs are expensive to maintain and ventilation is more easily controlled in unconnected drifts.

At some mines where haulageways were connected in loops, the custom was to leave single empty cars under the chutes to be loaded while the locomotive went out to the shaft. On the return trip the locomotive picked up these loaded cars one at a time, pushing them ahead of it, and left empty cars in their places. If chutes are properly constructed, better practice is to load the whole train at as few chutes as possible without uncoupling cars; and, if this is done, connecting loops in the loading drifts are unnecessary. This system is especially effective at mines where switching locomotives are used in loading cars and making up trains, and larger main-line locomotives haul the trains to the shaft.

Unless traffic is dense, single-track drifts are to be preferred, turnouts being provided at proper intervals. If double-track haulage is desired, unless the ground is unusually strong, to drive duplicate rather than double-width drifts may be more economical in the long run. The excavation cost for double-width drifts is not much greater than that for single drifts, but this may be more than counterbalanced by the higher cost of timbering and maintenance. Careful planning of turnouts or sidings will usually make double-tracking unnecessary.

In opening a new mine one may be undecided with respect to the system of mining that may be adopted with the best advantage, or conditions may be such that a certain system of mining may be indicated for the immediate future, to be followed later by another system. Under these circumstances the main-level development should be planned as far as possible in such a way that it will be effective for either system. This applies to level interval and to the position and spacing of main drifts, crosscuts, and raises, and may often be extended to sublevels and other intermediate development.

The cross-section of the drift depends on the size of the haulage equipment. A clearance of 18 in. on each side of cars and locomotives is advisable, and 1 ft. may be set as a minimum for mechanical haulage at places where travel is at full speed. Gable-buttomed, rocker-dump, and Granby cars of 60- to 70-cu.ft. capacity require drifts 8x8 ft. in cross-section, and 90- to 120-cu.ft. cars should have 9x9-ft. drifts. End-dump and box-type cars of equal capacity do not require as large drifts.

The 16- to 20-cu.ft. end-dump and rocker-dump cars most often met in hand tramming are disproportionately high and narrow, and were apparently designed to permit their passage through narrow drifts rather than to afford an opportunity for economical shoveling. For these cars drifts 5 ft. wide and 7 ft. high are sufficiently large.

The radii of curves in main drifts and crosscuts designed for mechanical haulage should be such that standard frogs and switches can be used, and are dependent on the gage of the track. The standard track gage for locomotive haulage in metal mines is 30 in., and the minimum safe radius of curvature for a 6-ton locomotive of this gage is 30 ft. Curves as sharp as this should

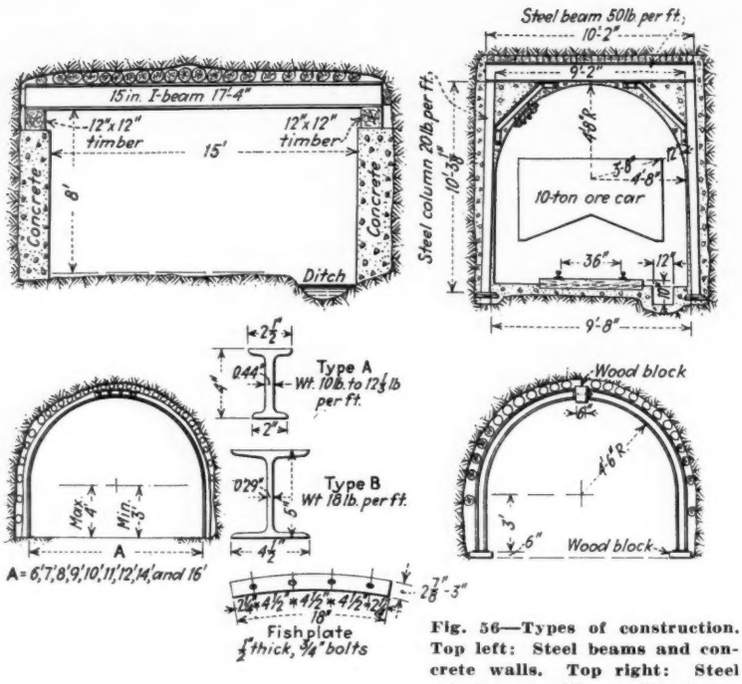


Fig. 56—Types of construction. Top left: Steel beams and concrete walls. Top right: Steel sets for use with or without concrete. Bottom left: British steel arch. Bottom right: Cushioned steel arch

be used only in emergencies, and a radius of 45 ft. is better for turnouts, and of 80 ft. or 125 ft. on the main line, where full speed is maintained.

Although in some drifts and crosscuts of long expected life the use of steel sets or concrete for ground support may be economical, in most instances where support is needed timber is best. The timber should be treated with a chemical preservative if, on account of decay, untreated timber would have to be replaced during the life of the drift. Drift sets have been practically standardized as to shape and design of framing, but not as to size. Nearly always the legs or posts should have a batter of 2 or 2½ in. to the foot. This batter is particularly necessary if much side pressure exists, or if the water ditch is large or deep. Vertical legs are likely to break down the edge of the ditch or to be pushed into it from the side. The ditch should be sufficiently deep so that the top of the water is lower than the bottom of the track ties, but this is a condition seldom realized in wet mines.

Size, shape, inclination, position, and spacing of chutes and raises depend to a large extent on the mining system and on the character of the ore, but certain principles govern their design and arrangement, which apply under nearly all conditions. Unfortunately, in our nomenclature, seldom is distinction made between an ore pass and the structure erected at its bottom to control the flow of ore, both being called chutes indiscriminately.

The size of the raise depends on the use to which it is to be put and on the ground through which it is to be driven. Small raises used as chutes have a tendency to hang up, especially untimbered raises, the walls of which have not yet been worn smooth. This tendency is sometimes overcome in round, vertical raises by decreasing the diameter towards the top. Square or oblong sections have been found to give less trouble on this account than circular or oval ones. If the chute has a turn, the section should be enlarged at this point.

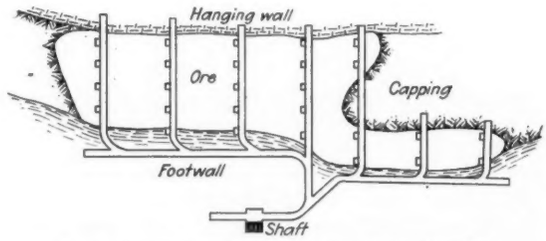
If timbering is not needed, and if there are several chutes, it is better to provide a separate ladderway and timber road than to partition off separate compartments in the chute raises, but this is not the case with timbered raises, customarily put up with two or more compartments. The usual size of untimbered, single-compartment raises is 5 to 6 ft. square, and of timbered raises 4 to 5 in the clear. In two-compartment raises the ladderway is often made narrower than the chute, but the tendency at present is toward making both compartments of the same size.

In timbered raises the length of the cribbing should be such that it can be cut from standard lengths of timber without waste.

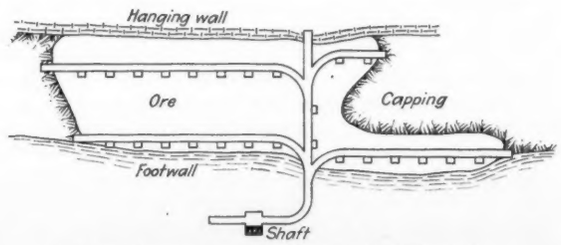
In general, the inclination of a chute or ore pass should be between 55 and 75 deg. For ore that runs freely and does not pack, the inclination may be as low as 50 deg., and if the ore is not allowed to accumulate in the chute it may in extreme circumstances be as low as 40 deg. Fifty-five degrees is an excellent angle for ore that runs freely, and about 70 deg. is best for ore that has a tendency to pack. If the raise is too flat, the ore will build up on the footwall. If too steep, it will pack and hang up. Vertical raises are particularly bad in this regard.

For reasons of safety, good practice should avoid putting up a raise in such a position that its top will be in the direct line of a drift, unless it is thoroughly protected or guarded. For the same reason a ladder road should not be placed in such a position that a man climbing down it will be endangered by passing cars. For these reasons, and for convenience in chute building, raises should be started from the side of the drift where additional excavation has been made, not from the back.

The spacing of chutes is important, and depends largely on the mining system used. If the chute itself is expensive, the raise above it may be branched for the sake of economy in construction, so that one chute will serve two or more raises. This may lead to complications if more than one grade of ore is to be mined, or if waste is to be kept separate, unless other chutes for the different grades are provided. If min-



A. Development by Crosscuts from Footwall Drifts



B. Development by Drifts from Central Crosscut

Fig. 57—Diagrams of underground development

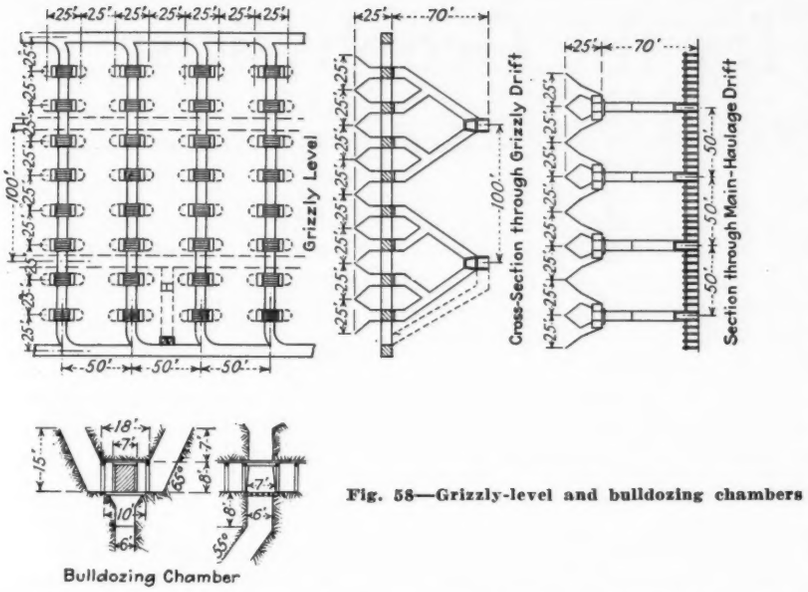


Fig. 58—Grizzly-level and bulldozing chambers

ing is done on contract, better results will be obtained, also, if each gang has its own chute.

The economic limit for direct shoveling is 13 ft. Consequently, in stopes where the ore is shoveled directly into the chutes, the spacing on top is usually about 25 ft. This is also the usual limit for chutes without branches under shrinkage or sublevel stopes. Here the distance between chutes is a compromise between the first cost of chute construction and raising, on the one hand, and the amount of ore tied up in pillars and hung up between chutes, on the other. In top-slicing and sublevel caving, the use of scrapers instead of sublevel cars has tended to reduce the distance between raises. Raises are now put up as close as 30 ft. apart, with a working radius of little more than 100 ft. on the sublevel.

In the undercutting and caving systems of mining, the spacing of the chutes depends on the spacing and arrangement of grizzlies and pull holes on the sublevels, and on the height of the grizzly level above the haulage level. This height depends, in turn, on the amount of storage room desired below the grizzly, and the degree to which the branching of raises is carried. The distance between chutes varies from 25 to 100 ft., and is usually less than 40 ft.

Some economy results in excavation on the main level if chutes are placed opposite each other in the drift, and this is commonly done when systems of branch raises are laid out, but it should not be done in heavy ground.

If one of the "wholesale" systems of mining is used, and if grizzly-level or bulldozing chambers are necessary for secondary blasting, the size, spacing, and arrangement of the grizzlies will depend on the spacing, number, and arrangement of the pull holes that converge at each end of each grizzly. The spacing of the pull holes usually varies from 12½ to 25 ft., with occasional variations beyond these limits. Common practice is to have two or even three pull holes converge at each end of the grizzly. Opinions on this point, however, are changing, and better practice now dictates use of a single pull hole.

If the grizzly level is 25 ft. or more above the main level, all the grizzlies and bulldozing chambers are usually connected by a sublevel at this elevation, to which a separate ladder road is constructed. If the grizzlies are at a lower elevation, the chambers may not be connected with each other, but each may have its own ladder road. Convenience is assured, especially in the matter of ventilation, when the grizzly level is independent of the haulage level, but this fact should not outweigh a considerable increase in the cost of excavation. In some mines the grizzly level is used for the distribution of timber and other supplies, and as a passageway for the men. This has the advantage of safety in keeping the men out of the haulageways, and it prevents the distribution of timber from interfering with ore traffic. Under the conditions noted, the grizzly level is either connected directly with the shaft or is connected with the haulage level by an incline, so that timber trucks may be taken up to it without unloading.

Grizzlies are commonly made of old rails, weighing from 50 to 90 lb. per yard, set in an inverted position. Material of this kind has been used because it could be cheaply and easily obtained from the railroads, but special shapes show less wear, and steel with a lower carbon content withstands better the shock of blasting and the fall of heavy chunks. Heavy I-beams, the upper flanges of which are protected by manganese steel straps or by inverted channels, have been successfully used, and standard bars are designed specially for this purpose. The width of the opening between bars varies from 10 to 20 in., and is usually 12 to 14 in.

■

Minerals in Tanganyika Territory

THE 1931 report of the Geological Survey of Tanganyika Territory, by Dr. E. O. Teale, is devoted in the main to a consideration of soil erosion, morphology and tectonics, and economic geology. In respect to the latter, the minerals discussed are coal, limestone (and cement), magnesite, gypsum, marble, diamonds, nickel, corundum, apatite, phosphate rock, graphite, and mica. Coal occurrences discovered thus far are limited to streaks and very thin coaly seams in carbonaceous and bituminous shale. Limestones, both primary and secondary types, are abundant. A few European producers use kilns of comparatively modern design, but most of the lime used is obtained from non-European contractors who use primitive methods of heap-burning. Roman cement, or pozzuolana, appears fairly widespread and includes fine clays, fine volcanic tuffs, volcanic dust, diatomite deposits and the like, which by simple admixture in finely ground condition with slaked lime and water become hydraulic cements. Magnesite occurs in several places, a porcellaneous, white, compact, cryptocrystalline variety in irregular veins and pockets in serpentine lenses being found at Mwahanza. Gypsum is found in four localities, scant information being available. Marble of varying grades is known in three areas. In the search for diamonds, several kimberlite occurrences have been discovered, but as yet few stones in such association. A pocket in the Mabuki region has become unprofitable after yielding 83,000 carats. Other pockets are sought. Magnetite rich in nickel has been found as float at Mwahanza, but, *in situ*, only narrow and discontinuous shattered veins have been disclosed. A sample of pyrrhotite carrying 0.53 per cent nickel is said to have come from Kimamba. Report of an occurrence near the east shore of Tanganyika at Kungive Bay has not been confirmed. Several finds of corundum, in biotite schist, have been made. An apatite (phosphate) marble, near Kisaki, is included among the marbles previously mentioned. Moreover, a limestone (sample), discovered at Galappo, has been found on analysis to contain 17.5 per cent tricalcium phosphate. Graphitic gneiss, schist, crystalline limestone, and marble occur, but no graphite of economic value.

Geology for the Geophysicist

Three Reviews by Dr. C. A. Heiland

HANDBUCH DER GEOPHYSIK, Band IV, Lieferung 3: *A. Sieberg: Erdbebengeographie (Seismo-geography). Berlin: Gebrüder Borntraeger; IV + 318 pp., illus. Single copy, Rm. 84; subscription price, Rm. 56.*

This volume of the Handbuch represents a most complete treatise not only on the geographic distribution of earthquakes, but on their history and relation to geologic structure. In the introductory paragraphs a review is given of the methods of investigation, and of the world distribution of structural elements of macroseismic importance. Seismic conditions of Europe, Asia, Africa, the Indian Ocean, Australia, the Pacific Ocean, North America, Central America and West Indies, South America, and of the Atlantic Ocean are analyzed. A geologic and seismic section through North America, and charts illustrating the seismicity of the Eastern part of the United States, of the southern Rocky Mountain region, of California, Mexico, and Cuba are of interest.

The author has subdivided each region into structural provinces and has represented their seismic activities in charts that demonstrate the distribution of earthquakes according to destructiveness and in relation to fault lines, block faulting having been recognized as more likely to cause earthquakes than all other possible geologic agencies. In addition, complete historical information is given in regard to the occurrence of earthquakes in every country, some of the data going back as far as 2000 B. C.

To present the material on seismogeography in such exhaustive form required a tremendous amount of statistical work and an analysis of the information on about 10,000 earthquakes. The practical importance of such work is threefold. It affords access to data in regard to seismic and related phenomena for all regions in all parts of the civilized world where earthquakes have occurred in the past, and tells how destructive they have been, so that steps can be taken accordingly in the design of buildings and structures. Secondly, the records covering the distribution and intensity of earthquakes often furnish valuable additional information in an analysis of the regional geologic conditions of an area; finally, as earthquakes occur where tensions are released that have been accumulated by disturbances of the equilibrium of the outer crust, data on earthquakes are in many instances related to disturbances of isostatic equilibrium as disclosed by gravity observations.

The patience and care applied by the author in the preparation of this volume

will make it second to none as a reference book on seismo-geography for a long time to come.

HANDBUCH DER GEOPHYSIK, Band IX, Lieferung 1: *B. Gutenberg: Der Aufbau der Atmosphaere (Constitution of the Atmosphere). B. Gutenberg: Die Schallausbreitung in der Atmosphaere (Propagation of Sound in the Atmosphere). J. Tichanowski and R. Muegge: Waermehaushalt der Stratosphaere (Radiative Equilibrium in the Stratosphere). Berlin: Gebrüder Borntraeger; V + 171 pp.; illus. Single copy, Rm. 36; subscription price, Rm. 24.*

The physics of the atmosphere will be treated in the Handbuch in two volumes (VIII and IX). The material to be reviewed is the first part of the second volume. In its first section, Gutenberg discusses the composition of the atmosphere in the lower layers, and the results of observation of temperature and of wind directions. He then deals with the phenomena observed in the stratosphere, from which conclusions may be drawn, in regard to its constitution, such as the luminous night clouds, zodiacal light, meteors, Kennelly-Heaviside layer, the aurora, and the green line in the auroral spectrum. This is followed by a discussion of the theoretical investigations into the composition of the atmosphere from thermodynamic considerations, from the behavior of meteors, and from the ionization as deduced from the Heaviside-layer observations. The last chapter then presents a picture of our views on the constitution of the atmosphere from the combined experimental and theoretical results as described in the foregoing, chiefly in regard to temperature, chemical constitution, and density.

The composition of the atmosphere also influences the propagation of sound; and, because of the increase of velocity with altitude, the sound rays are curved in much the same way as the seismic rays in the earth interior, which explains the zone of silence observed in explosions (and, in a related manner, the curvature of short radio waves around the globe). After a brief discussion of the theory of sound-recording apparatus, the relation between pressure, temperature, and sound velocity, the calculation of the travel-time curve and of the curvature of the sound rays are treated. The last chapter deals with actual observations of sound propagation, in particular with the zone of silence, echo observations, travel-time curves, angles of incidence, and the velocity of sound in the stratosphere.

The temperature at any point of the stratosphere, in which convection currents and condensation phenomena are absent, is controlled by two currents of energy flowing in opposite directions. The inflow is due to the sun's radiation and that of the atmospheric strata; the outflow is due to the reflected terrestrial radiation and the upward radiation of atmospheric strata. In the section written by Tichanowski and Muegge, the various theories dealing with the constituents of the radiations mentioned, and the radiation equilibrium as a whole, are discussed in detail.

HANDBUCH DER GEOPHYSIK, Band II, Lieferung 2: *A. Born: Der geologische Aufbau der Erde (Geologic Structure of the Earth). Berlin: Gebrüder Borntraeger; V + 303 pp.; illus. Single copy, Rm. 69; subscription price, Rm. 46.*

A tremendous amount of well-correlated geologic information has been compiled in this volume, which is a veritable compendium on the regional geology of the world. The author deals first with the structural geology of Europe, then follow Asia, Australia and the Pacific Ocean, Africa, North America and Greenland, Central America and the West Indian Archipelago, South America, Antarctica, and finally the Atlantic Ocean. These areas are divided up into regional geological units; not a mere description of the geologic conditions is given, but genetic relations are emphasized wherever possible, and data on the structural history, such as orogenic and epirogenic movements, resulting character and thicknesses of formations, and occurrences of mineral deposits, are given. A great number of geologic maps and sections are included, in which the arrangement of structural elements such as folds, synclines, massives, faults, sedimentary troughs, and other geological phenomena is demonstrated. Frequent references are made to the latest geological literature.

Corresponding to the much greater amount of data available on the regional geology of Europe, the chapter on the structure of Europe occupies about one-half of the book. However, the geologists of North America would probably have liked to see the geology of their continent treated in more detail, as more data on its regional geology

are available than about Africa or Asia, to which almost the same space has been devoted.

This book should be of considerable value to the geophysicist. Many phenomena in the field of gravity, earth's magnetism, seismology, and isostasy require correlation with the regional geology, and many if not most geophysicists would not know where to find the desired information. Furthermore, the geophysicist engaged in practical work will find this book to be of particular value. In the interpretation of anomalies found by geophysical methods, there is an increasing tendency to view results obtained locally in the light of the regional geology, and a book of this character should be of great assistance in promoting a better type of interpretation.

In addition, in this book will be found quite a few references to the regional structure of areas in which geophysical work has been done. An incomplete list of these follows: district of Kursk, the Emba field, the Siegerland, the Carboniferous basin of Westphalia, the Hartz Mountains, the Bohemian mass, Upper Silesia, the Black Forest, Central France, the Guadalquivir basin, the Southern Alps, the Carpathian Mountains, the Pannonian basin, Roumania, the Basin of Paris, North German salt anticlines, Persia, Mesopotamia, Dutch East Indies, North Africa, Rhodesia, Pennsylvania, basement uplifts in Texas, Oklahoma fields, Gulf Coast salt domes, Californian ranges, and other geological phenomena. Finally, the structural maps of the North and South American continents will also be of interest.

It is difficult to do justice, in a brief review, to the amount and quality of work done by the author in compiling this material, and to the cooperation of the publishers in reproducing the many geological maps and sections.

GEOLOGY. *By W. H. Emmons, G. A. Thiel, C. R. Stauffer, and I. S. Allison. New York: McGraw-Hill Book Company; pp. 514, illus; $4.*

If the question be asked as to why another textbook on geology should be published, reply may be made by pointing to the rare incident of four experienced teachers of geology collaborating to produce a textbook. Of fact or theory the volume contains nothing new. But in the excellent selection, arrangement and presentation of the textual matter and the illustrations, the book can hardly be surpassed as a medium for imparting knowledge of the elements of geology to first-year college students. The topics of the seventeen chapters following the introduction are: Nature of materials that constitute the earth; weathering; the atmosphere; ground water; gradational work of streams; gradational work of snow and ice; lakes and marshes; the ocean; sedimentary rocks; diastrophism; vulcanism; mountains, their origin and structure; metamorphism; structure of rocks; mineral deposits; probable conditions within the earth; and, a brief outline of earth's history.

FLOTATION PLANT PRACTICE. *By Philip Rabone. Pp. 141; illus. London: Mining Publications, Ltd.; 10s. 6d.*

A series of lectures delivered by the author on modern flotation practice has been compiled in book form. They are not intended for the flotation expert but are designed to give a broad concept of the subject with no more detail than is necessary for the purpose; cost and capacity figures needed for preliminary calculations in the field have been included.

To keep the content within reasonable limits, description of out-of-date equipment has been omitted; processes and flowsheets have been selected with the same consideration. Almost half the text has been devoted to crushing, grinding, and classification technique prior to actual flotation treatment. Much space is also devoted to the discussion of flotation reagents. No bibliography has been included. Those wishing detailed information on the subject are referred to more advanced works.

UNTERSUCHUNGSMETHODEN FÜR ROHEISEN, STAHL UND FERROLEGIERUNGEN. *By J. Kassler. Pp. 158; illus. Stuttgart, Germany: Ferdinand Enke Verlag; Rm. 19.60.*

Drawing from extensive experience in modern steel plants the author has selected the most reliable and up-to-date analytical methods available in present steel metallurgy and alloy technique. No attempt has been made to include other methods than those of proven value, and complicated procedures have been omitted whenever more rapid and equally exact methods have been found available. The gravimetric determinations of chromium and manganese, to give an example, have not been included, as the present rapid volumetric methods have been found equally good.

An interesting detail is the description of the microchemical determination of carbon, manganese, chromium, and nickel in steel.

LASSETER'S LAST RIDE. *An Epic in Central Australian Gold Discovery. By Ion L. Idriess. Ninth edition; 251 pp., illus., index. Sydney, New South Wales; Angus & Robertson; 6s.*

In this journal for December, 1931, and October, 1932, appeared reviews of Idriess's second and third editions of "Prospecting for Gold," in Australia. He appears to be a popular writer in that country, because five editions of a story of an inland mission have been printed, and also one edition on northeast Queensland and three editions on Gallipoli and Palestine.

Idriess is a practical and descriptive writer. He knows Australia. "Lasseter's Last Ride" is not fiction; it is a splendidly presented tale based on the records of the Central Australian Gold Exploration Company, and on the related experiences of members of the expedition. The story in brief is this: Thirty-two years ago, L. H. R. Lasseter reported that he had discovered rich gold ore in central Australia, near the South Australian-Western Australian border. He became lost, and an Afghan camel driver found him raving with thirst and clutching a bag of specimens. A surveyor took care of him, and three years later the two re-located the claim, but on returning to the outside world they found their time incorrect; therefore, the bearings of the vein were in error. However, in 1930, Lasseter persuaded some Sydney men to put up money, and the company mentioned was floated. A party of five experienced men was equipped with two motor trucks, a wireless outfit, an airplane, and ample supplies also. They set off from Alice Springs, a hamlet in the center of Australia. But the remainder of the story is one of trouble with the trucks and airplanes, anxiety about water, thick bush, hills, sand, natives, delays, failure to detect the country traversed originally, and doubts; and finally Lasseter's decision to go on alone with camels. To one of his men he said: "Well, so long. If I don't find the reef [vein] I am never coming back."

And he did not, although his body was eventually found and buried. His wanderings and the danger he faced when with the blacks (aborigines) have been pieced together from fragments of letters buried by him beneath the ashes of his campfires. It was a fruitless and futile expedition of well-equipped men; but in July, 1932, discovery of gold was reported at what is now known as the Granites, near the border of the two states, and considerable exploration is under way by similarly equipped parties. Perhaps Lasseter's original find may be rediscovered. Let us hope so. Truly the prospector is a pioneer of civilization. M. W. VON BERNEWITZ.

MINING DIRECTORY—CATALOG AND ENGINEERS' HANDBOOK. *Los Angeles, Calif.: Los Angeles Chamber of Commerce; $2.*

A compilation, by the members of the mining committee of the Los Angeles Chamber of Commerce, of data, tables, and charts taken from reliable sources, against which "International Critical Tables" was used for checking. In addition to articles on the principles of mining and on mining law, the volume contains much information of value to mining engineers, metallurgists, and geologists.

Personal Notes . . . Comment . . . Criticism

A. B. Watson has returned to Nigeria from England.

E. R. Scott is now a member of the staff of the Department of Geology, University of North Carolina, Chapel Hill, N. C.

C. W. Wright, chief engineer of the mining division of the U. S. Bureau of Mines, is now chairman of the Washington, D. C., section of the A.I.M.E.

Harold B. Wright, formerly of Kalgoorlie, Western Australia, and of Prestea, West Africa, has returned to England on the completion of metallurgical work in Spain.

Sherwin F. Kelly addressed the Science Forum of the New York Electrical Society on Dec. 14 on the subject of "Exploring Downwards—The New Science of Geophysics."

Arthur S. Hecht, contributor of the article on "Microscopic Determination of Ore-Treatment Methods" in this issue, was, until recently, metallurgist with the Ringwood Company, Ringwood Manor, N. J. Earlier engagements included several years as mining engineer with copper companies operating in Arizona. Mr. Hecht is a member of the eastern magnetite mining and milling methods committee of the A.I.M.E.

P. M. Newhall recently spent a vacation of four months at his home in Berkeley, Calif., returning to his post of general manager for the North Venezuela Petroleum Company, at Puerto Cabello, Venezuela, via New York and London. Mr. Newhall was formerly prominently associated with underground developments in the Witwatersrand, South Africa, particularly in connection with the introduction of the jackhammer drill.

Noel H. Stearn, author of the article, in this issue, on "Mining and Furnacing Quicksilver Ore," graduated from Stanford University in 1917 and was awarded the degree of Ph.D. from the University of Wisconsin for a thesis on the application of geomagnetics to exploration. Following field seasons with the Wisconsin Geological Survey, Doctor Stearn was appointed instructor in the department of geology of the University of Wisconsin. He then undertook professional consulting and exploration work in the Iron Country (Michigan, Minnesota, Wisconsin), Mexico, British Columbia, Arkansas, Arizona, Missouri, Tennessee; and acted as a member of the Science Advisory Committee of the National Research Council. Doctor Stearn, who co-invented, with Dr. W. O. Hotchkiss, president of the Michigan College of Mining and Technology, the superdip magnetometer, is now chief geologist for Southwestern Quicksilver.

Tellurides in Canada

The Editor:

In the article on "Developments in Flotation-Cyanidation," in your December issue, there is a description of a flotation-cyanidation process, evidently that of the Lake Shore mine, in the Kirkland Lake district, and in connection with it the article contains this statement: "In spite of the fact that this concentrate contains a high percentage of gold tellurides, the extraction is about 98 per cent." Would it be a fair question to ask what evidence there is concerning the existence of this high percentage of gold tellurides? For thirty years or more there has been loose talking about the behavior of tellurides in cyanide solutions, based on assumptions rather than any real knowledge. Our work on ores from some of the other mines in the Kirkland Lake district shows that the gold-bearing tellurides therein are fairly readily dissolved in the ordinary milling practice. It is high time that the presence of tellurides as an excuse for high tailings and as a recognition of the value of peculiar processes should be challenged on every occasion. H. E. T. HAULTAIN.

University of Toronto,
Toronto, Ont., Canada.

C. W. Dowsett, until recently general manager for Bwana M'Kubwa Copper, operating in Northern Rhodesia, has returned to Canada to take up his new duties of consulting metallurgist to Dome Mines, South Porcupine, Ontario.

Allen Cross (left), aviator and manager, New Guinea Airways; with Tom Harris, field superintendent for Bulolo Gold Dredging.

Frank F. Espie, of Burma Corporation, has returned to Australia on furlough.

J. G. Farrell, on the staff of Anaconda Copper in Chile for 24 years, is on a holiday visit to Australia.

Irving B. Crosby, of Boston, is the author of a report on the mineral resources of Massachusetts, recently issued by the Massachusetts Industrial and Development Commission.

Gee Hong Geng, a graduate in mining engineering of the New Mexico School of Mines (1923), now on the staff of the engineering department of the arsenal at Nanking, China, and **Col. Chang Lien-Ko,** a graduate in mining engineering of Tokio University, now a member of the Ordnance Research Commission and the Ministry of Military Administration, of Nanking, China, are in the United States making a study of copper smelting methods.

Ernest N. Patty, dean of the Alaska Agricultural College and School of Mines, writes that last summer was the first time students from the school found summer employment difficult to obtain. Most of them had work in sight before the summer vacation began, and several had to create new jobs. The first two to hire themselves were Fred Parker and Harold Schmidt, sophomores. They leased some placer ground on Moose Creek in the Fairbanks district, repaired an old ditch, sluiced off some of the overburden, set up boxes, and began "shoveling-in." They were fortunate in finding high-grade pay, which gave them good wages. At the end of the first month Schmidt left to accept another job, so Parker took on a second partner and continued working until college opened in the fall. The boys cleaned and melted their own gold and the bars were sold to the local bank. The ground which they worked was a small block on a well-known creek left by early operators. A few miles distant from the Parker-Schmidt placer mine another venture was under way. Hugh M. Henton, associate professor of metallurgy, and two of his students, Fred Kubon and William Burns, took a lease on the stamp mill tailing of the Crites & Feldman mine. During the summer they installed and successfully operated a small plant. This was the first cyanide plant to be installed in interior Alaska, and their pioneering created much interest. The plant was mostly assembled from equipment from an old brewery in Fairbanks. Five wood-stave tanks served for leaching and solution storage. Beer kegs proved effective as zinc boxes. Wheelbarrows were used for delivering the tailing into the solution tanks. Less than half of the tailing was treated during the first season. The work will be resumed next summer.

Origin of Pedestal Rocks

The Editor:

A letter of mine, published in the Aug. 23, 1924, issue of *Engineering and Mining Journal*, referred to the so-called Anvil Rock, which, prior to its dislodgment by an earthquake in 1915, stood on the southwestern slope of Cinnabar Mountain, in Pershing County, Nev. In that letter I predicated my ideas concerning the origin of the pedestal rock on erosion. I now wish to correct my earlier statement with an announcement, based on subsequent findings, that may also account for the existence of pedestal rocks elsewhere.

Artifacts found in the deserts and old lake bottoms of Nevada have revealed a system of measurements that show Star Peak, in the Western Humboldt Mountains, to be the apex on an angle of a triangle, the adjacent sides of which extend for specific and related distances. The apex of the angle to the southeast falls on the spot formerly occupied by Anvil Rock. Investigation also shows that an old trail, formerly used by the Mayas who inhabited the region, passes through the valleys and foothills, in a true north-south direction, near the site of the rock. Inasmuch as the rock could be seen for many miles by those using the trail, it was an excellent landmark.

The side of the triangle connecting Star Peak with the site of Anvil Rock is exactly 26 miles long. This is significant. The number 26 is twice thirteen, the number of lunar months in a year. Similar facts of a mathematical and astronomical character, derived principally from measurements of the faces of the rock and from a study of the history of the former inhabitants of the region, indicate conclusively that Anvil Rock was fashioned and erected by the Mayas.

By applying the information gained as a result of the aforesaid studies, I have already discovered several other pedestal rocks in Nevada, all of which have occupied, in each instance, a specific position in a triangular system. The many mathematical and astronomical features of these systems are particularly interesting, but their description is too involved to be presented here.

Lovelock, Nev. JOHN T. REID.

W. L. Whitehead, consulting geologist for Utah-Apex Mining, was recently examining mines in the Mother Lode and Nevada City-Grass Valley districts of California.

Jesse L. Maury makes an important contribution to the subject of mine accounting in a recent bulletin (Vol. XIV, No. 4) of the National Association of Cost Accountants.

W. J. Turner, who has had extensive managerial experience in New South Wales and Western Australia, has been appointed mine manager of the White Hope South mine, at Cracow, Queensland.

Paul R. Cook (Misssouri School of Mines) announces from Moscow his intention to return to gold mining in some other part of the world after the termination of a contract with the U.S.S.R. now approaching completion.

Dr. E. C. Ulrich, paleontologist for the U. S. Geological Survey, recently received, at a meeting of the American Geological Society, the Penrose medal. The presentation was made by Dr. W. O. Hotchkiss, president of the Michigan College of Mining and Technology. Dr. Ulrich's discoveries have added two geological systems to the geologic column—the Ozarkian and the Canadian, lying between the Cambrian and the Ordovician.

W. S. Chapman, analyst and assayer, South Australian Department of Mines for 41 years, will retire shortly.

Obituary

John U. Sebenius, retired chief mining engineer of Oliver Iron Mining; at Rochester, Minn., after an operation; on Dec. 18; aged 70. Mr. Sebenius was born in Sweden and educated at Stockholm. After association with the Witherbee-Sherman company he practiced as a consultant in Duluth, contributing to the geological knowledge of the Iron Country and taking part in the development of metallurgical processes for the concentration of Mesabi ores, as exemplified in the concentrator at Coleraine.

E. V. Graybeal, chief mining engineer for Miami Copper; of a self-inflicted shot, by accident, while on a hunting trip in the Mazatzal Mountains near Miami; aged 40. Mr. Graybeal, a graduate of the Colorado School of Mines, had been connected with Miami Copper for the past twelve years, most of the time as chief mining engineer. Mrs. Graybeal, who survives, with two sons and a daughter, is a daughter of Dr. E. P. Mathewson, of the faculty staff of the University of Arizona.

A. S. Bandler, president of Bahia Corporation, specialist in black diamonds; at New York on Dec. 28; aged 59. Mr. Bandler, also president of Bernard Bandler & Sons, black diamond merchants, was granted in 1925 a 30-year concession from the government of Bahia, Brazil, to work diamond-bearing ground. As an outcome, the Bahia Corporation was formed in 1927.

Robert F. Segsworth, prominently identified with the mining industry of Ontario; at Toronto on Dec. 18; aged 65. A barrister by profession, Mr. Segsworth was owner, with his brother, Walter E. Segsworth, of the Seneca Superior mine at Cobalt, and a director of Dome Mines.

E. P. Clark, assistant counsel of Title Guarantee & Trust, New York; at Brooklyn, N. Y., on Dec. 12; aged 72. Mr. Clark was graduated from Columbia School of Mines in 1880, practising thereafter for several years in Colorado as a mining engineer.

Edward Heberlein; in Paris on Dec. 22; aged 52. Doctor Heberlein was well known as a mining and metallurgical engineer in many parts of the world.

Stephen S. Tuthill, secretary of the American Zinc Institute from 1918 to 1929; at Brooklyn, N. Y., on Dec. 1; aged 63.

Wilson Bradley, co-inventor of the Bradley-Laury process for leaching manganiferous iron ores; at Deerwood, Minn., on Dec. 9.

J. W. Revere, great-grandson of Paul Revere; at Canton, Mass., on Dec. 11; aged 84. Mr. Revere was graduated from the Massachusetts Institute of Technology in 1868, and was associated throughout his entire professional career with mining in eastern Canada and Nova Scotia.

Alex Wise, a prominent figure in the mining industry of Nevada for 30 years; at Reno, Nev., on Nov. 8; aged 52. Mr. Wise was graduated from the University of California in 1900. In 1919 he reopened the Imperial and Yellow Jacket mines, at Gold Hill, on the Comstock Lode, and sold his options to United Comstock. He operated the Flowery Mines, near Virginia City, on low-grade ores, establishing a record for low operating costs. His most recent venture was the purchase of the Beebe gold property, at Georgetown, El Dorado County, Calif., where he installed the first commercial Hadsel mill, described in the issue of *Engineering and Mining Journal* of December last. Two weeks before his death, following an attack of influenza, he sold the property to the Bradley interests, of San Francisco.

Gotthold Koelle, director of Metallgesellschaft; at Frankfort-on-the-Main, Germany, on Nov. 21, 1932; aged 58. Educated at Zurich, under Lunge, Doctor Koelle was later associated with Dr. Ferdinand Heberlein, one of the inventors of the Huntington-Heberlein process, and with the firm of Huntington, Heberlein & Company. In 1901 he joined the staff of Metallurgische Gesellschaft, sponsors of the process, and took charge of its introduction at Penarroya, at the Harz, and elsewhere. When Metallurgische Gesellschaft introduced the Herreshoff furnace, Doctor Koelle was entrusted with the work. He also assisted in the introductions of other inventions, notably the Wedge furnace, the Dwight-Lloyd process, and the Cottrell process. He was appointed a regular member of the board of directors of Metallgesellschaft in 1917.

John H. Mackenzie; at San Francisco, Calif., on Dec. 23; aged 75. Mr. Mackenzie's career was marked by varied achievements in mining and management, and he earned among his contemporaries a high position of respect for his ability and personality. He was born in Canada, ran away to sea at 13, and started mining in Nevada in 1877, becoming active in mining ventures there and in Idaho, California, Arizona, and Montana subsequently. In 1898 he joined the Alaskan gold rush and organized the Klondike Mines Railway. From 1903 to 1905 he was associated with F. W. Bradley and Mark L. Requa, in consulting work. In 1907 he was appointed manager of Goldfield Consolidated's operations in Nevada. Active association with Bradley and Requa was resumed in 1910, in connection with the development of the Alaska Juneau property. Tungsten mining claimed his attention during the War.

INDUSTRIAL PROGRESS

Clean-Up Speeded in Small Cyanide Plant By New Zinc-Dust Precipitation

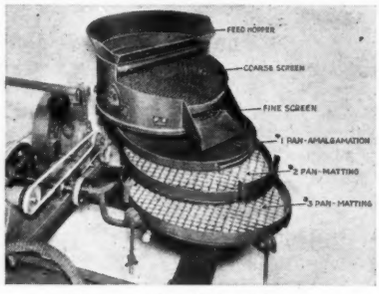
A new development in the precipitation of gold and silver from cyanide solutions, which is intended primarily for use in small cyanide plants treating 50 to 200 tons of solution daily, has been announced by the Merrill Company, San Francisco. The combined equipment used for clarifying, de-aërating, and precipitating is said to have been simplified and cheapened without lessening the efficiency of the process. The clean-up, it is claimed, is quicker and cleaner than with any form of zinc-dust precipitation previously used.

A single centrifugal pump effects the successive steps of clarification, de-aëration and precipitation. The larger of two rectangular tanks is kept filled to a constant level with unclarified solution, the inflow being controlled by a float valve. Vacuum clarifying leaves with outlets connected to a manifold are hung in the tank. The manifold in turn is connected to the top of a vertical de-aërating tower. A second pipe connects the top of the tower with a vacuum pump, the pipe being carried high enough to form the usual barometric seal. Within the tower the solution passes down through suitable screens or grids, which break up the flow into small streams and films, effecting a substantially complete removal of dissolved oxygen.

The clarified, de-aërated solution is withdrawn from the bottom of the tower by a vertical single-stage centrifugal pump, submerged in solution to prevent re-entry of air through the pump gland. If the pump be located at a suitable height above the bottom and its capacity be ample for the required solution tonnage, the tower will neither empty itself nor fill above a certain height. This arrangement obviates the need of any form of float control within the tower and permits the use of simpler, smaller, and cheaper towers.

Zinc-dust emulsion is introduced to the pump suction midway between pump and tower. Continuous and uniform feed of zinc dust to the emulsion cone is supplied either by a belt-type feeder or by a revolving-drum feeder. Precipitated barren solution is supplied to the cone by a pipe connected to the smaller or barren solution tank, and the mixture of zinc dust and solution is discharged from the cone by a mechanically operated valve, which is opened and closed by a cam and adjustable tappet operated by the zinc feeder, thus permitting uniform introduction of zinc emulsion. Less than 1 g.p.m. of oxygen-free barren solution is discharged by this valve.

The clarified de-aërator solution, now containing the proper amount of zinc dust, is forced by the centrifugal pump into cylindrical pressure filter bags suspended in the barren tank. Precipitated solution flowing from the bags into the tank overflows through a weir and thence to barren solution storage. The filter units consist of an outer bag of heavy canvas and an inner one of sheeting. Clean-up is effected by raising the bags, blowing with air, and removing the inner sheeting bag containing the gold-silver precipitate. After adding flux, the sheeting bags are rolled up and burned in the melting pot, followed by the usual fluxing and melting. If preferred, the bags may be washed and reused, in which case the washings are filtered and the precipitate is melted. The former method involves no handling of precipitate.



New mechanical gold pan

Panning Gold Mechanically

A mechanical gold pan has been devised by the Denver Equipment Company, Denver, Colo. It is shown in a cut on this page. It consists of a nest of pans, 24 in. in diameter, which is given a panning motion by being tilted slightly and shaken by means of a steel yoke and saddle connected to a rotating eccentric. Power is provided by a ¾-hp. gasoline engine, large enough to drive a pump, also, for furnishing any necessary water. The first concentrating pan, immediately below a fine screen, is of copper coated with amalgam, to catch principally the finer gold. This pan discharges by gravity to the second pan, directly below, which is of steel having rubber matting in the bottom, covered by heavy wire screen, to save the heavy gold. The screen gives a swirling action to the mass. This pan discharges into a third one, of the same construction, which recovers the remaining gold and any floured mercury.

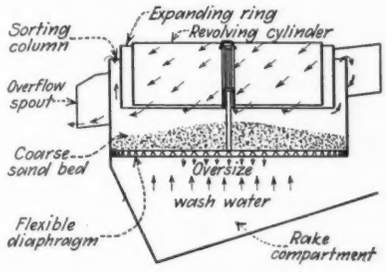
Another Separator for Placer Gold

A gold-concentrating device developed by C. J. Whitlock, San Bernardino, Calif., consists essentially of a vertical cylinder or drum made of sheet metal in which revolves a perforated steel disk, which is double, having a bottom floor which slopes at a variable angle. This disk is attached to a hollow revolving shaft. The gravel is fed wet from a trommel screen into the top of the separator, falling on to the disk which revolves at 150 r.p.m. The gold and other heavy minerals are separated by centrifugal force and precipitated through the perforations in the upper plate. Tailing passes over the outer edge of the disk, and, encountering a circular baffle plate below the disk, is deflected back to the vicinity of the central shaft, where it is dropped to approximately the center of the bottom chamber of the machine, on the concave floor of which a rake revolves which effects further separation, the heavier material being retained and the lighter rejected as final tailing. A bumper below the drum causes it to drop sharply a half inch at every revolution. The concentrate passing down through the tubular shaft is collected below. The machine is run by a 2-hp. engine and has a rated capacity of 50 tons daily.

Refinement of Cone Classifier

A cut on page 39 shows the so-called Auto-Vortex classifier, developed by Charles E. Wood, Baxter Springs, Kan. This is a refinement of Mr. Wood's automatic conical classifier, used in the Tri-State district. Novel features include: Use of a feeder spout which discharges the feed tangentially into a revolving cylinder, from which it passes through ports in the bottom into the uprising current of the sorting column and over the overflow rim into the launder, from which it is discharged at one side. The uprising velocity in the sorting column is regulated by adjusting the diameter of an expanding ring, which varies the sectional area of the column. Wash water is introduced in a

vortex through which the bed material must pass when necessary. Machines of this type in commercial use are handling up to 100 tons per hour of oversize with 1,500 g.p.m. feed water and making an undersize product as coarse as eight mesh, while overflowing as



This automatic conical classifier embodies new improvements

much as 75 per cent of the feed tonnage. A segmented flexible diaphragm is used to form a bottom for the sorting column. This will support a bed of coarse material a few inches deep, but, if the bed becomes excessive, each segment will bend and allow some of the bed to drop into the rake compartment. Wash water pulsates upward through the diaphragm and the material bed, washing the fines back into the uprising current.

For Metallographers

A photographic plate designed especially for photomicrography of metal specimens has been developed by the Eastman Kodak Company. The new product is known as the Wratten metallographic plate and has been sensitized with a new dye especially suitable for this work. The characteristics of the plate are: High resolving power; availability of a very high photographic contrast; availability of lower contrasts; and high green sensitivity as well as high sensitivity to blue light; also absence of sensitivity to red light, which permits freedom in darkroom handling.

High-Chromium Steel Economical in Rabble Arms

A mining company in South Africa, engaged in roasting lead and zinc ores, is reported to have standardized on high-chromium cast steel for roasting-furnace rabble arms and teeth that are subjected to severe operating conditions. Roasting temperature is often as high as 1,470 deg. F. Arms and teeth are also subjected to the corrosive action of furnace gases containing sulphur dioxide and to the abrasive action of the ore. The company now specifies chromium-nickel steel containing 27 to 30 per cent chromium and 2 to 3 per cent nickel with a maximum of 0.15 per cent carbon for all rabble arms and teeth, and a straight 15 per cent chromium steel for the lute rings and the hearth rings. Such use of high-chromium steel is reported to have resulted in a saving in operation and maintenance. Rabble arms made of it should, it is said, last almost indefinitely, and the teeth should have five to forty times the life of ordinary cast iron. The 30 per cent chromium 3 per cent nickel alloy does not ordinarily require cooling, this eliminating heat losses by cooling water or air, and furnace destruction by water in case of leak. Moreover, furnace capacity is increased and the cost of the cooling system is eliminated. The greater life of the alloy also results in fewer shutdowns and less fluctuation in operation.

A New Use for Rubber

Rubber is useful in various ways to obviate noise. The B. F. Goodrich Company, Akron, Ohio, cities an instance where a $\frac{3}{4}$-in. hydraulic pipe in one of the company's buildings developed a noisy water-hammer due to fluctuating pressure. No remedy could be found until an engineer substituted a piece of $\frac{3}{4}$-in. hose for a section of the pipe. This ended the difficulty.

Oil-Proof Cord

Portable cords and cables having a special outer sheath impervious to the destructive action of oil and grease have been introduced by the Okonite Company, Passaic, N. J. These are intended for service where such cords are exposed to contact with oil, resulting in the deterioration not only of the outer covering but eventually of the insulation itself.

Fine Reduction Cone Crusher

A new development of the Symons cone crusher has been announced by Nordberg Manufacturing Company, Milwaukee, Wis. It is known as the "short-head fine-reduction crusher" and has much in common in appearance and principle of operation with the Symons standard cone. The chief differences are in the shape and the crushing cavity and the design of the head and the bowl. The head is shortened, the diameter of the intake opening at the top of the crushing cavity being greatly increased. This greater area allows much of the initial crushing to be done at the top. The steeper angle permits faster passage and is said to be an important factor in securing the large capacity claimed. The parallel zone at the bottom has been made longer, assuring proper sizing. The new machaine is built in sizes having 3-, 4-, $5\frac{1}{2}$- and 7-ft. diameter cones.

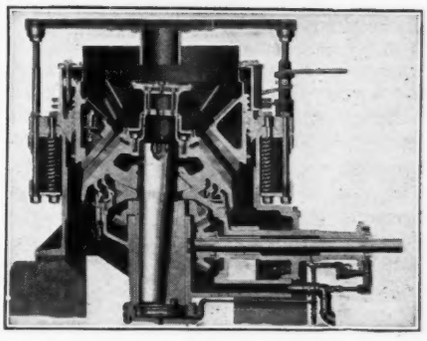
Speed Reducer Drive

A motor reduction unit for low-speed drives has been introduced by Allis-Chalmers Manufacturing Company, Milwaukee. It comprises a standard round frame-type motor mounted directly on the gear-reducer housing. Helical gears are used.

Surveying Bore Holes

United States Patent No. 1,877,593, dated Sept. 13, 1932, has been granted to Irwin Roman for a method and apparatus for surveying bore holes. Dr. Roman is assistant professor of mathematics and physics at the Michigan College of Mining and Technology. The device is mounted in a large tubular casing so made that it may be lowered into the hole. In the casing bottom are mounted an incandescent lamp and a lens which cause parallel rays of light to pass up through the tube. In the ray path are fixed two magnetic compasses, one near the bottom of the instrument and one near the top, in such a way that the needles are always horizontal. A photographic device in the top of the casing, its axis coinciding with that of the casing, records the position of the needles as the tube is lowered. The needles differ in pattern so that they may be distinguished on the record. Because they are always horizontal, and the axis of the camera is parallel to the drill hole axis, they will appear shorter as the hole direction deviates from vertical. Inasmuch as the ratio of apparent length of the compass needle to its true length equals the cosine of the angle of deviation from vertical, and as the compasses provide a means for determining true North, the course of a drill hole may thus be completely surveyed.

Two compasses are used to make possible correction for local attraction. In surveying, readings are first taken when the top compass is at surface. The instrument is then lowered until the high compass is where the lower one was. Readings are then made again.



A cone crusher for fine reduction work

A Novel Filter Press

A filter press that by a slight change can be converted into a press exerting a pressure of over 500 lb. per square inch has been developed by the Thompson Continuous Filter Press Company, Seattle, Wash. Features are said to include continuous operation, continuous filtration through the filter fabric, and low friction losses. Automatic control and variable speed are provided.

Portable self-priming sump pump for 10- to 40-ft. lifts.

Portable Sump Pump

An open-impeller type of centrifugal pump driven by a multi-vane air motor, both inclosed in a one-piece housing, has been introduced by Ingersoll-Rand Company, New York, for service as a sump pump. The new unit is portable and air-operated. It weighs 50 lb. and is designated as Size 25. It is intended for lifts of 10 to 40 ft. with 70- to 90-lb. air pressure. It is self-priming, if submerged to cover the inlet screen.

Improved Safety Headgear

A novel protective hat, designed for miners and known as the Cool-Cap, is here illustrated. It has a safety crown formed and ribbed from pressure-molded moisture-proof fiber, and is ventilated through nine openings. Visor and lamp holder are of cowhide strongly sewed,

the former being designed to carry an electric or a carbide lamp. A soft leather head-rest and a felt-lined conforming head-band promote comfort. The cap is light and permits vision through 180 deg. It is made by the Portable Lamp & Equipment Company, Pittsburgh, Pa.

Substitute for Rubber

A new chemical product, said to have practically all the characteristics of rubber but to be unaffected by solvents as rubber is, has been given the name of Thiokol. It is an olefinpolysulphide reaction product. Among various applications made or suggested for it are: protective covering for underground low-voltage cables; high-voltage cable-covering in place of lead for protection against soil and corona conditions and strong current; ignition cables to protect against oil and corrosive conditions; and for protecting dropcords against oil in machinery.

Photo-electric Relay for General Use

Increasing use of the photo-electric relay for controlling the operation of many kinds of equipment makes of interest the new general-purpose photo-electric relay recently introduced by General Electric. Among its advantages, the new relay affords more sensitive control than older models and permits operation at a light intensity as low as three foot-candles. This is only 30 per cent of the minimum light intensity prescribed for older models.

Arc Quenched With Boric Acid

Boric acid is used in a fuse designed by Westinghouse to break 20,000 amp. at 13,200 volts without arcing. In the center of a thick-walled cylinder of solid boric acid is a plunger, which would be withdrawn deep into its hole by a spring, were it not held forward by the fuse-wire which serves as one terminal. When the fuse melts, the arc is drawn down into the boric-acid hole, and elicits an outrush of water vapor which instantly extinguishes it. The fuse is of the De-ion type.

■

Industrial Note

Whitcomb Locomotive Company, Rochelle, Ill., subsidiary of Baldwin Locomotive Works, has acquired the business, inventory, and good will of Milwaukee Locomotive Manufacturing Company, under a plan whereby Westinghouse Air Brake Company, formerly the owner of Milwaukee Locomotive Manufacturing Company, becomes a stockholder in the Whitcomb Locomotive Company.

■

BULLETINS

Crushers. Allis-Chalmers Manufacturing Co., Milwaukee, Wis. Bulletin 1472. Superior McCully gyratory crushers. Pp. 24.

V-Belt Drives. Dayton Rubber Manufacturing Co., Dayton, Ohio. Catalog 105. Dayton cog-belt drives. Pp. 32.

Instruments. Lucius Pitkin, 47 Fulton St., New York. Leaflet. The Tour temperature controller.

Instruments. Bailey Meter Co., Cleveland, Ohio. Bulletin 300. Fluid meters, for steam and water. Pp. 24.

Air Transport. Junkers-Flugzeugwerk A.G., Dessau, Germany. Pamphlet entitled "Aerial Freight Service in New Guinea." Pp. 7.

Motors. Reliance Electric & Engineering Co., Cleveland, Ohio. Bulletin 112. Fully inclosed fan-cooled induction motors, Type AA, Form F.

Transformers. Wagner Electric Corp., St. Louis, Mo. Bulletin 172. Air-cooled transformers.

Electric Welding. Metal & Thermit Corp., 120 Broadway, N. Y. Catalog. Murex heavy mineral-coated welding electrodes.

Crusher. Hadfields Ltd., Sheffield, England. Bulletin No. 346. Stone crusher and ore crusher, built entirely of steel. Pp. 16.

Track Supplies. Hadfields Ltd., Sheffield, England. Bulletin No. 345. Hadura manganese-steel switch point. Pp. 4.

Pumps. De Laval Steam Turbine Co., Trenton, N. J. Catalog B-6. Propeller pumps. Pp. 4.

Laboratory Sieves. Newark Wire Cloth Co., Newark, N. J. Folder. Standard testing sieves.

Electric Welding. Harnischfeger Corporation, Milwaukee, Wis. Bulletin HW-2. Electric welding machine. Pp. 20.

Dredge Buckets. Hadfields Ltd., Sheffield, England. Leaflet No. 341. Johnson bucket-pin locking device. A new contrivance for securing the pins in dredge buckets and preventing their movement when buckets are in use.

Conveying. Diamond Rubber Co., Akron, Ohio. Pamphlet on conveying problems entitled "How Shall We Handle It?" Pp. 24.

Welding. Linde Air Products Company, New York. Pamphlet No. 25-M. Bronze patented welding rod for bronze welding. Pp. 20.

Plant Upkeep. Lead Industries Association, New York. Pamphlet entitled "The First 2,000 Years," a record of white lead paint performance. Pp. 12.

Paints. New Jersey Zinc Co., New York. "Using Paint as Light," and ABC summary of "The Influence of the Reflecting Characteristics of Wall Paints Upon the Intensity and Distribution of Artificial and Natural Illumination," by D. L. Gamble, Research Division.

Diesel Engines. Atlas Imperial Diesel Engine Co. Catalog No. 932-F, on stationary diesel engines and calling attention in particular to the company's application in mine service. Pp. 32.

Corrosion Resistance. Electro Metallurgical Co., New York. Pamphlet, "Stainless Steels and Their Uses." Pp. 20.

Lubrication. National Carbon Co., Cleveland, Ohio. Catalog. Sections G-132, G-232 and G-332 on Gredag lubricants.

Compressors. Sullivan Machinery Co., Chicago. Catalog No. 88-A. Improved angle-compound compressors. Pp. 32.

Motors. General Electric Co., Schenectady, N. Y. Catalog. Synchronous motors. Pp. 52.

Chain Drives. Morse Chain Co., Ithaca, N. Y. Bulletin No. 51. Roller chain drives. Pp. 56.

Nickel Cast Iron. International Nickel Co., New York. Abstract. Guide to selection of engineering specifications for gray iron castings.

Drying, Pulverizing. Raymond Bros. Impact Pulverizer Co., Chicago. Leaflet. Raymond kiln mill.

Excavators. Bucyrus-Erie Co., South Milwaukee, Wis. Leaflet. Red Arch Type-V dragline bucket.

TRENDS AND DEVELOPMENTS IN THE INDUSTRY

NORTHWEST

Dividends From High-Grade Silver. Sunshine Mining (Coeur d'Alenes), which recently paid a dividend that brought total disbursements to $1,043,292, is one of the few silver mining companies that has continued paying dividends at present low metal prices. The mine, near Kellogg, Idaho, recently opened up a vein, on the 1,900-ft. level, that contained ore assaying from 700 to 800 oz. silver per ton. . . . Idaho Mineral Products, under the management of J. L. Mercer, is rehabilitating the Vienna mine, a gold-silver-lead property near Hailey. The camp was closed in 1893 and reopened under the present management in 1930. Sufficient ore has been developed to justify the remodeling of the mill, for the production of concentrate by flotation. A 200-hp. diesel engine and generator is in operation, and supplies are on hand sufficient for the winter. Output is expected to be resumed early in 1933. . . . F. M. Handy, of Spokane, representing a syndicate, has bought Phir Gold, occupying more than a mile and a quarter of shear zone near Elk City, Idaho, and near the Gnome mine, recently a gold producer. From 10,000 to 15,000 tons of ore is developed and partly blocked. This is expected to be more than doubled by the work planned prior to the erection of a mill. The new owners plan to do 600 to 800 ft. of work before June 1 in development to be started soon, and then to install a 40-ton unit of a mill that will be increased to 120 tons' daily capacity as development justifies.

Association Convention a Success. Charles H. Goodsell, of Spokane, was re-elected and three new members have been elected to the board of trustees of the Northwest Mining Association. The three new members are Compton I. White, Sandpoint, U. S. Representative-elect; Milnor Roberts, Seattle; and Frank Woodside, Vancouver, B. C. Hold-over members of the board are Robert T. Banks, president; L. K. Armstrong, vice-president; F. Cushing Moore; Leon Starmont, secretary; D. M. Drumheller, Jr.; Charles Hussey and Frank M. Smith, all of Spokane; and Donald A. Callahan, Wallace; A. E. Drucker, Pullman; Eugene White, Tacoma, and Dr. Francis A. Thomson, Butte. The association held its annual convention in Spokane, Dec. 15, 16, and 17. An international economic conference was urged for the earliest possible moment, and resolutions adopted discouraged efforts to remove the currencies of the world from their metallic base by the substitution of greenbacks. Included in the resolutions was one urging that the Bureau of Mines be transferred to the Department of the Interior, and another recorded opposition to any legislation that might affect a severance between surface and subsurface rights. Further legislation in connection with mine depletion was also urged. An interesting exhibit at the convention was a working model of a placer gravel washing machine invented, designed, and built by F. Cushing Moore, mining engineer, of Spokane, and former president of the association. The machine is designed for use on ground not workable by any other method and on a tract too small to justify the expense of dredging.

ALASKA

Activity North of Fairbanks. Idaho-Alaska Corporation (Harris Hammond, president; W. R. Wade, vice-president and general manager; Hon. John Hays Hammond, chairman of the board), has taken over the leases and options held by Engineers Exploration Syndicate on various gold properties in Chandalar, Alaska, and is starting development work. This district, 240 miles north of Fairbanks, has several 4- to 6-ft. veins exposed by shallow workings, the average oreshoot sampling running $35 to $73 per ton. Airplanes will be used to transport miners and supplies into the district this winter. E. A. Boadway is local manager in Alaska. . . . Idaho-Alaska Corporation has also taken over the interests of Engineers Exploration Syndicate in Silver City, Idaho, and will start opening the gold mines on War Eagle Mountain. R. S. McClellan is local manager in Idaho. W. R. Wade was recently in Idaho in the interests of Idaho-Alaska Corporation.

Alaska Juneau Profits. During November the Alaska Juneau company mined and trammed to the mill 339,920 tons, recovering gold valued at $260,500, or 76.62c. per ton. Operating expenditures totaled $171,500, or 50.45c. per ton, and operating profit, $89,000. The directors recently placed the stock on an annual dividend basis of 60c. a share. A quarterly dividend of 15c. a share, payable on Feb. 1 to holders of record on Jan. 10, was voted on Dec. 22, 1932.

OKLAHOMA

Zinc Smelter Resumes. The smelter at Blackwell, Okla., operated by Blackwell Zinc, subsidiary of American Metal, recently resumed production and gave employment to about 230 men. The plant, which has 4,000 retorts, suspended operations in November, 1930. R. C. Weisbrod is general superintendent.

CALIFORNIA

Widespread Activity in Gold. Martinez Gold Mines, operating near El Dorado, reports the strike of a new orebody in the main shaft, which is being sunk an additional 150 ft. Assays are said to range from $15 to $25 a ton. . . . The Cleveland group of gold claims in the Bullychoop district, 42 miles west of Redding, has been taken under bond and option by a group of California and Oregon capitalists. The old workings are to be rehabilitated, and the lower tunnel extended 400 ft. to intersect the main orebody. Construction of a 100-ton mill is contemplated. . . . Work of installing a 50-hp. West Coast oil engine, compressor, hoist, and pumping equipment is proceeding at the Sailor Boy mine, near Mojave. A. F. Carper is in charge of operations. . . . Installation of a diesel power unit, a 200-ton mill, and other equipment has been completed at the King Solomon mine, near Yreka. Production is to start at once. The camp is being stocked with supplies, to permit operation during the winter. . . . Loftus Blue Lead Mines announces that operations are to start at its property on Howland Flat, near Downieville, with the coming of the wet season. Installation of new equipment has just been completed. Abundant water is now available at many points of the property. . . . Greenwood Gold Mining continues development of the Gilt Edge mine, in Placer County, with good ore reported. The property formerly was known as the Revenge. . . . Paramount Mines Corporation has increased its holdings in the Ophir district by purchase of the Bradbury group. New equipment is to be added to increase production and recovery. The mill is operating steadily on $30 ore, producing about $3,500 a month.

Plant Changes and Mine Developments. Rehabilitation of the Glenn gold mine, at Mokelumne Hill, and reconditioning of the twenty-stamp mill, is being undertaken by Raldo Exploration, of Los Angeles. . . . Idaho-Maryland, at Grass Valley, is operating at full capacity. The company recently completed a program of plant modernization, involving an expenditure of $200,000. . . . The Empress mine, at Newton, which suspended production to facilitate deepening of the main shaft, is now mining and milling 60 tons a day. The shaft is now at a depth of 800 ft., with 1,000 ft. as the objective. Work on a 200-ton mill is to start soon. . . . Production will be stimulated at the Vallecito Western Gravel mine at Six Mile Creek, in Calaveras County, with the sinking of a 2,000-ft. shaft by Tonopah-Belmont Mining, which acquired the property recently. The new shaft will serve to develop additional ground known to contain pay gravel. Opera-

Pumice deposit at Cobre, Nev., being developed by Cobre Minerals Corporation. The material finds many uses, particularly for admixture with concrete.

tions are in charge of Fred. Bradshaw, chief engineer of Tonopah-Belmont. . . . American Development, of New York, has acquired the Bunker Hill gravel mine near Gibsonville. . . . Two pumps of 600-g.p.m. capacity have been ordered by Empire-Star Mines, for installation on the 1,600-ft. level at the Murchie mine, near Nevada City. The pumps will serve to convey the water to the 1,300-ft. level, whence six 4-stage multiplex units will raise it to the surface.

Bradley Interests Acquire Beebe Properties. Control of the Beebe gold properties at Georgetown has been acquired by the Bradley Syndicate, operating several mining enterprises in California and Nevada. Milling has been temporarily discontinued, and all activities are centered on developing substantial ore reserve on the lower levels. The main vein at a depth of 250 ft. is 50 ft. thick and averages $4 a ton. Operations will be carried on by Atolia Corporation, a Bradley subsidiary. . . . Carson Hill Gold Mining has reconditioned more than 4,600 ft. of the main tunnel at the Carson Hill property, and is making arrangements to remodel the 700-ton mill of thirty 1,250-lb. stamps. The ore, of which a considerable tonnage is said to be in sight, is to be mined by glory-hole methods. . . . J. W. Mullin, of Beverly Hills, has acquired control of the Mazeppa and Jumper mines, near Sonora, and is driving a tunnel 2,300 ft. to open up the properties. Over 275 ft. has been driven to date. Three shifts are employed. Ore obtained from the tunnel face and stoping operations is being treated at the Mazeppa mill. . . . A new ball mill and flotation machinery are among improvements installed at the Sliger gold mine in the Georgetown district. The main plant is treating 50 tons of ore a day. . . . Oregon Creek Company, headed by O. A. Johnson, of Sacramento, plans to resume operations at its True Gist property near Comptonville, after a two months' shutdown. . . . Exploitation of a portion of the Hespie Dam gravel channel near Gibsonville, in Sierra County, is being started by Brown Consolidated Gold. The company has been working placers northeast of Gibsonville for about four years. The new holdings acquired recently will be worked by hydraulic methods. . . . The old Chapman mine near Railroad Flat is being reopened by Sacramento interests. Erection of machinery and installation of pumps has commenced. George Buyck is in charge of operations. . . . Five feet of gold-bearing gravel has been encountered in the two-compartment shaft now being sunk on the Broder ranch, near Falsom, by Gray Wing Extension Mining, according to E. R. Gray, superintendent. Additional test holes revealed that the discovery will add $250,000 to the tested value of the bed. An extensive mine surface plant, including a mill and hoisting and pumping equipment, was completed recently.

NEVADA

Gold Mining Continues Active. Atlas Gold Mining, Edgmont, Elko County, has completed installation of a 50-ton flotation mill, to operate during the winter as a test plant. An 18-in. redwood pipe line 4,700 ft. long has been laid, and an electric hoist installed. The mine was operated from 1902 to 1908 by Montana Mining, of London, England, and was closed by litigation over a Montana property owned by the same company. H. H. Carpenter, mining engineer, formerly of Los Angeles, Calif., is president and manager. . . . Monarch Gold Ledge Mines will install 120-hp. Fairbanks-Morse diesel engine and electric generator at White Rock, 50 miles east of Fallon. Electric power will be transmitted to the mine over a 3-mile line. The company has also purchased equipment for a 50-ton amalgamation and flotation mill, to be built at White Rock. R. A. Fraser is general superintendent. . . . Gold Ore Mining, Gold Road, Ariz., has taken a lease and option on the Ramsey-Comstock gold mine at Ramsey, 35 miles east of Reno, from Lahonton Mines. The property was acquired on the recommendation of Albert Burch, consulting engineer for Gold Ore Mining. Victor Light, a director of the company and assistant manager of Tom Reed Gold Mining, Oatman, Ariz., will have charge of development. The Ramsey-Comstock mine is under water to the 400-ft. level. . . . J. L. Dobbins, of San Francisco, has purchased the McTigue custom mill, at Silver City, on the Comstock Lode, and is installing a ball mill and Kraut-flotation plant. The ball mill will relieve the stamp mill and permit increase of capacity of the unit. Two other custom mills, the Donovan and Trimble, are milling ores, and Comstock Silver Mining is operating a stamp mill on company and lessees ore. Bullion is sold to the United States assay office, formerly the mint, at Carson City, 12 miles from Silver City. . . . Gold Circle Consolidated, at Midas, Elko County, expects to complete unwatering of the Elko Prince gold-silver mine on Jan. 1, and will then resume operation of its cyanide mill. The working shaft is 750 ft. deep and a winze extends to the 900-ft. level, in ore averaging $50 a ton for an average width of 15 in. The 750-ft. level and the winze have been under water since 1922, when the cyanide mill of Elko Prince Mining burned. Noble H. Getchell is general manager, and F. W. Varney superintendent.

MONTANA

Gold-Mining Interest Predominates. Little Ben Mining, operating the August mine near Landusky, Phillips County, recently received a 100-hp. Fairbanks-Morse diesel from Chicago. It was hauled by truck a distance of about 1,300 miles, and delivered, in less than three days. The engine will soon be in operation, fueled with Montana crude oil, according to advices received by Montana Stock & Bond. Property has been shipping gold ore returning from $7,500 to $14,400 per car of 40 tons. . . . Walter Mallette, and Fred Callaway, of Spokane, and T. T. Pohlman, of Spokane, have purchased the 50-ton mill at the High Up mine, near Virginia City, Madison County, and are moving it to their Prospect property in the same district. . . . Nelson Story estate, operating a placer mine in Norwegian Gulch, near Norris, Madison County, will shortly start its new floating dredge, with buckets of ½-yd. capacity. . . . Discovery of $25 gold ore near the vicinity of the Ermont mine started a stampede in that district, all the claims being located for miles around. The district is about 13 miles west of Dillon and the same distance east of Bannock, Beaverhead County. . . . Butte Copper Consolidated's lead property, known as the Joe Dandy group, near Radersburg, Broadwater County, is shut down until lead prices return to normal. The company is working a group of gold properties in a new section of the Basin district, Jefferson County. . . . W. F. Pauline, Butte business man, and associates are operating a group of gold claims near Iron Rod, Madison County. . . . . The mill of the Callahan Gold Mining, east of Jefferson City, Jefferson County, recently burned to the ground, entailing a loss of over $20,000, not covered by insurance. The stamps and crushers will be used in a new plant ordered. A ball mill will be added, giving the new plant a capacity of 100 tons per day. Meanwhile, shipments of ore are going forward to the smelters, under supervision of Owen F. Brinton. . . . Dr. Francis A. Thomson, president of the Montana School of Mines and head of the Montana Bureau of Mines and Geology; John D. MacKenzie, manager of the East Helena smelter of the A. S. & R.; Clinton M. Roos, head of the Vosburg Syndicate, Helena; D. Anderson, of the Golden Gate Mining, Iron Rod; and Carl J. Trauerman, president of Montana Stock & Bond, Butte, represented Montana at the annual meeting of the Northwest Mining Association, held in Spokane Dec. 15, 16, and 17.

ARIZONA

Nevada Company in Arizona. Tonopah Mining, of Tonopah, Nev., has acquired the Hidden Treasure mine, in the Harqua Hala Mountains, 15 miles southeast of Salome, in Yuma County, and is moving in equipment from Tonopah in preparation for development work. George Meyers, a prospector from Yuma, found gold on the property on a contact between limestone and quartzite. He turned his claims over to Ed. (Cactus) Johnson, of Salome, who in turn interested the Tonopah company.

Zinc Oxide Process Developed. Golconda Mining is constructing a 50-ton mill 18 miles north of Kingman, in which a new process for making zinc oxide, patented by John Allingham, will be used. The ore contains zinc, copper, lead, gold, and silver, but zinc predominates. Sam Black, of Kingman, is president of the company, and those in charge of operations at the property are R. O. Gruwell, Jr., and J. A. Gruwell. The company has been developing the property and testing the ore in a small pilot plant for three years. Aside from the Golconda, few of the base-metal properties in the district south and east of Chloride are active, and since the Kingman Chloride branch of the Santa Fe railroad was abandoned a year or so ago, the little town with a chemical name has taken on the appearance of a ghost camp.

Placer Progress. The placer districts of Mohave and Yavapai counties are receiving attention from numerous itinerant prospectors, and several small companies have started operations. Not fewer than 150 prospectors are reported to be running dry washers in the gulches around Rich Hill and on Weaver Creek 15 miles north of Wickenburg. These men operate their little hand-operated dry washers on any location that appears promising to them. Few make more than 50c. or a dollar a day, and placer-property owners figure that the prospectors may find for them a bonanza or that their presence may help to sell the property. In this district John N. Walters has 22 men working on the Gowdy claims, owned by the Lion Mines, of Los Angeles. A derrick and a Leroy gasoline hoist are used to hoist the gold-bearing gravel to a small bin which discharges into a small cylindrical concentrator which separates out the gold by centrifugal action. The property totals 60 acres and lies on the western outskirts of Octave. . . . Placers in the Gold Basin and Lost Basin mining districts, in Mohave County, north of Kingman and a few miles south of the Colorado River, are coming into prominence. Kingman residents estimate that 75 dry washers operate in the two districts, which lie in adjacent townships, and two companies are testing ground and preparing to install placer machinery. They are Four Square Mining and Gold Basin Development. Other Mohave County placer districts that are being prospected either by individuals or by companies are the Silver Creek district, which is between Oatman and Goldroad, and the Chemahuey's district, which is approximately south of Topock.

COPPER COUNTRY

Cone Crushers to Replace Stamps. Copper Range is planning to replace steam stamps with cone crushers. The crushers will be electrically operated and will handle more ore at less cost. These expectations are based on the results of tests. The problem presented by malleable copper in the rock has been solved by removal of the metal freed by crushing and by use of crushers in series. The innovation is another step to enable the Michigan mines to compete with copper properties in other parts of the country. Cheap power is provided by the Copper District power plant in which Copper Range owns a half interest. . . . In so-called "normal" times, average cost to produce a pound of copper in the Michigan district was 13c. Up to 1928, for a period of five years, the average profit was 0.885c. Most of the companies were able to pay dividends on even so small a margin, so the future, with assured lower costs, is considered not so dark as present conditions seem to indicate. Costs should not average more than 10c. per pound when copper is again sufficiently in demand to bring a fair price. Calumet & Hecla and Copper Range, with the introduction of economies in labor, power, materials, and methods, may do even better when normal operating conditions again prevail. Both companies have a particular advantage in cheap power; Calumet & Hecla has a modern, low-cost steam plant. Under normal conditions Quincy should produce copper at about 10c. per pound. When operations ceased because of low-price metal, Quincy had long stretches of lode heavily charged with copper. This ground was opened in the last two years preceding the shutdown. When work was stopped, the company was prepared to produce and handle a larger tonnage than ever before, at lower cost. Only a limited number of stopes had been started, the reserves remaining intact. . . . Mohawk concentrate, stored near the stamp mills at Gay, is being smelted at the Michigan smelter, in Houghton, as shipments are received. Shipments will be continued throughout the winter if the railroad from Gay can be kept open. Mohawk still has remaining in storage about 12,000 tons of 70 per cent concentrate, according to trustworthy estimates. Much of the original stock has already been converted into refined copper. The Michigan smelter is smelting concentrate from Copper Range as well, with only occasional shutdowns for furnace repairs and incidental adjustments to equipment.

NORWAY

New Sulphur Process Developed. A new process for extracting sulphur from sulphide minerals has been developed by Orkla Grube-Aktiebolag, the largest Norwegian producer of pyrites. A subsidiary company, Orkla Metal A/S, has been formed to exploit the process, and a plant capable of handling 200,000 metric tons of pyrites annually has been erected at a cost of about four million crowns (a little over $1,000,000). This will give a yearly output of about 70,000 tons of sulphur, placing Norway among the important sulphur-producing countries of the world, according to "Tidsskrift for Kjemi og Bergvesen." Increasing competition from the sulphur producers long ago made the Norwegian pyrites producers realize the importance of developing a process for direct extraction of the sulphur in their ores. Experiments at Orkla date back to 1919, and the results obtained gave sufficient promise of success to justify the erection of a test plant for treatment of 40 tons of pyrites per day. This plant produced 3,157 tons sulphur in 1929 and 3,244 tons in 1930. In the new process the pyrite is first melted with coke, limestone, and quartz, to flux the main part of the iron. Copper and the precious metals combine with sulphur to form a matte, and the remainder of the sulphur is expelled in the furnace gases in free state, as sulphur dioxide and as sulphur-carbon compounds. By careful control of furnace gases and the use of proper catalysts, the sulphur dioxide is made to react with the sulphur-carbon compounds, forming carbon dioxide and free sulphur, which is condensed and collected. A small amount of sulphur dioxide left in the gas after the condensation is absorbed by scrubbing in a battery of six towers filled with limestone. The condensed sulphur is finally treated by a nodulizing process, which eliminates dust and produces a granular product said to be an excellent product for handling in the common types of mechanical roasting furnaces. It is of high purity, entirely free from bituminous matter, and has an ash content rarely exceeding 0.01 per cent. A recovery of 85 to 90 per cent of the sulphur and copper content is obtained. Heat recovered in cooling the furnace gases and the condensation of the sulphur supplies sufficient steam power to operate the plant.

Copper and Molybdenum Production. The new flotation plant at Kongens Grube, of Roros Copper, has been put in commission. Daily capacity is 150 tons. . . . The Knaben molybdenum mines at Fjotland are now operating at full capacity, employing about 150 men. A new rope tramway has been completed between mine and mill, with a capacity of 800 tons of ore per sixteen hours.

YUGOSLAVIA

Lead-Zinc Operations Profitable. For the year to Sept. 30, 1932, Trepca Mines' profit was £165,735, before providing for depreciation and taxes, compared with £157,467 in the previous yearly period. In October and November the company's over-all costs, excluding depreciation and taxes, were less than 15s. per ton of ore, against 17s. 4d. a year before. This does not include smelting, as the company sells its products as concentrates.

RUSSIA

U.S.S.R. Meets Obligations. Tetiuhe Mining Corporation, which sold its properties in Siberia to the U.S.S.R. government a year ago for a cash payment of £7,860 and State notes to the face value of £932,140, is receiving payments regularly. The notes are collected as they mature, and have thus been re-

duced already to £871,140. Payments are expected eventually to liquidate the company's loans and debentures, and return par for the outstanding shares. The sale was made at a price agreed upon by the engineers representing the U.S.S.R. and those acting for the company.

CANADA

Noranda's Status and Prospects. James Y. Murdoch, president of Noranda Mines, speaking to members of the Canadian Institute of Mining & Metallurgy in Ottawa, stated that the refinery in Montreal East, handling copper from Noranda and Flin Flon, is producing at the rate of 50,000 tons a year. From the unambitious prospecting company organized by S. C. Thomson and H. W. Chadbourne in 1922, Noranda has become in ten years the third largest gold producer in Canada, the 1932 output being $7,000,000 out of a total of $65,000,000. With copper output restricted by agreement, it still supplies the larger part of the 50,000 tons to the refinery. Dividends have totaled $9,600,000, and the liquid assets are $10,000,000. The mine, smelter, refinery, and rolling mill emloy 1,900 men, the payroll is $2,500,000 annually, and $3,000,000 a year is spent in the purchase of supplies.

Teck-Hughes Develops Read-Anthier. Teck-Hughes has started a campaign of deep-level development from Level 41 to Level 50, the latter to be at a depth of 6,105 ft. The company is meeting its dividends comfortably from the lower average grade of ore now available from the mine, and Levels 35 to 40 show an improvement in grade over those immediately above, but the present average may not persist into the undeveloped ground beneath. That the company should expend a part of its substantial surplus in the attempt to make a new mine is therefore logical. Its most promising prospect is the Read-Anthier property, in Bourlamaque Township, east of the Siscoe. Irregular quartz veins occur in a granodiorite-greenstone contact zone, with the erratic but promising values that have characterized several properties in this locality, notably Siscoe and Treadwell-Yukon. Diamond drilling to a depth of 400 ft. and an extent of 7,000 ft. has given inconclusive results, so Teck-Hughes has arranged to provide money for a 500-ft. shaft, from which a thorough testing of the deposits will be made. . . . Lake Shore's flotation-cyanide treatment has proved so satisfactory that it will be applied to the tailing dump accumulated in Kirkland Lake during the earlier years of operation. Construction of a 1,000-ton tailing plant has been started as an addition to the mill, which is estimated to give an annual output of $500,000, or $1.50 a ton, with about eight years' supply available.

Low Radium Costs Expected. Great Bear Lake now has a permanent population sufficient to warrant monthly mail flights, the first being completed in December. The radium reduction plant at Port Hope, Ont., is producing commercial radium salts, using the method for obtaining the radium-barium concentrate developed by the Department of Mines at Ottawa. With rich ore and with an extraction method considerably more efficient and less expensive than the processes of recovery in use elsewhere, it is expected that Canadian radium will be the cheapest yet produced. How far this has been attained in the initial stages of the plant at Port Hope remains to be seen. . . . Beattie Gold Mines has completed its mill buildings and has the larger part of the heavy mill equipment installed, with the remainder on the ground. Mining development continues to show better results than did the diamond drilling, with an average somewhat over $4 per ton. The source of hydro-electric power has not been determined, the Quebec Government having refused to permit the Ontario Hydro-Electric Commission to bring power into the province from its Abitibi system, where much cheap power is available; and the private power company serving the Rouyn field, near by, has established rates that are too high to permit profits from a low-grade ore such as Beattie will treat. . . . Ashley Gold Mines appears for the first time in the official gold bulletin of Ontario for the month of November. The first return was $22,000, and shipments have since been made regularly. The mill is running at about half its capacity of 150 tons, and the grade of ore has gradually been brought up to over $20.

Importance of Diamond Drilling. Underground exploration at Kirkland Lake during the past few months has brought remarkably good results. The latest development is in the western part of the Wright-Hargreaves ground, where faulting has previously caused a gap in the productive zone between the workings of this mine and those of its neighbor Lake Shore. Diamond drilling at the 2,500-ft. level has indicated substantial ore sections in this block, and drifting is under way to open them out. At the western end of the "main break" the persistent underground prospecting of Macassa continues to bring in a moderate amount of ore. . . . Equipment and supplies for Island Lake to the extent of 800 tons are now on hand at the northern end of Lake Winnipeg, including a 50-ton mill, mining equipment, and supplies for a year. This will be hauled across the 120-mile winter road during January, February, and March. Meanwhile, sufficient supplies are being flown in from the railway to start mill construction. Underground work will be deferred until the mill is ready to operate, probably in March. The ore in sight and indicated by diamond drilling is of high grade, and the operators of the property intend to measure the grade with the mill as development proceeds. . . . Central Patricia Mines has arranged to build a 50-ton mill at its property on Crow River, where a mining plant is already installed and 45,000 tons of $13.50 ore has been developed. This will be the most remote gold mine to operate in Ontario, being considerably farther from the railway than the Howey, to the west, though closer to the rails than Island Lake, in Manitoba.

■

BRITISH COLUMBIA

Steady Increase in Gold Output. A gratifying feature of mining operations in the Province has been the increase in gold production in 1932. The total value is estimated at $3,800,000, and a further increase to probably $5,000,000 is anticipated for 1933. At present 90 per cent of the lode gold is derived from ores of which gold is the sole or principal constituent. The impetus given to gold mining is evidenced by the considerable increase of shipments of custom ore to the Trail smelter, amounting to at least three times the tonnage consigned for custom treatment in 1931. Thus, in 1931 the total custom ore treated in Trail from British Columbia mines was 6,059 tons, of which 2,270 tons was gold ore. Shiments in 1932 up to Dec. 7 had amounted to over 20,000 tons, of which about 17,000 tons was gold ore. Silver production for the year is estimated to be about 7,000,000 oz., compared with 7,500,000 oz. in 1931. The price averaged about 3c. per ounce higher than in 1931, on account of exchange premiums. Placer gold yielded $330,000, compared with $290,000 in 1931. The returns do not indicate the full activity in this branch of mining.

Successful Development Stimulates Activity. Encouraging results attending operations of Cariboo Gold Quartz have given impetus to exploration and development of properties in the same area. Cariboo Consolidated Gold Mines has bonded the properties acquired recently by Reward Mining and has secured the necessary funds from American interests to carry out a development program. The properties include locations made in pioneer days, when, in 1865, a stamp mill was erected and ineffectual attempts were made to open up orebodies. During the past season Reward Mining has carried out exloration to develop the vein system at greater depth, and encouraging possibilities have been indicated.

AFRICA

Important Transvaal Development. A new company, East Daggafontein Mines, will take over the Government mining lease over the eastern portion of Farm Daggafontein No. 9, successfully tendered for by Rand Selection Corporation, in association with Central Mining & Investment and New Consolidated Gold Fields. The tender provides that if development operations in the lease area warrant further development, the program will include the sinking of an additional shaft to that taken over from Daggafontein Mines, and the erection of a reduction plant. If development operations do not justify this, the new company will have the right to apply for a further area adjoining the lease area if such area is then available. Alternatively, the new company will be entitled to select a portion of the lease area not exceeding 800 claims, in which to carry out more intensive development, and to abandon its lease over the remainder of the area.

N'Changa Orebody Is Vast. Rhokana Corporation's consulting engineer, H. S. Monroe, reports that the N'Kana ore horizon has now been traced a length of 19 miles, half of which is yet to be

tested by drilling. Referring to his estimate in 1931 that the N'Changa West lower horizon orebody contains 6,300,000 tons averaging 17 per cent copper, Mr. Monroe mentions the extent to which underground openings exceeded that estimate. An incline raise for 195.5 ft. averaged 27.5 per cent for a width of over 6 ft., and a footwall drive for 163 ft., 22.7 per cent over a similar width. Despite these indications in the vast N'Changa orebody, Sir Auckland Geddes remarked at a recent meeting that the property is being forgotten by the public, which is thinking of Rhokana now only in terms of the N'Kana mine. During the year 7,265 square miles of Rhokana's prospecting concessions was traversed and mapped, making 33,500 square miles completed. Of the seventeen new finds made, six of gold and one each of copper and graphite will be further investigated.

Encouraging Reports From Kenya. Rhodesia-Katanga has nearly completed the drilling of its Kansanshi property, in Northern Rhodesia. This is an area of rich veins and limestone-replacement deposits, unlike other Northern Rhodesia mines. Recent boreholes have disclosed some sensational assays. Rhodesia-Katanga is interested with Tanganyika Concessions in the Kakamega gold field, which the geologists and prospectors of these companies are now examining. Reports from Nairobi indicate that further promising discoveries of gold have been found in the new Kenya field, in an area about 4 miles from Eldoret and 8 miles in the direction of Kakamega. Sir Albert Kitson's interim report to the Kenya Government was followed by legislative action to oust natives from the area and to settle them elsewhere. A stampede of white prospectors has occurred, and inter-racial trouble is reported.

AUSTRALASIA

Company Promotion Active. Formation of many gold companies has been a recent feature of Australian mining. Every gold mine where operations may be carried on profitably with gold at £7/7/- per ounce is being unwatered, and prospecting is being undertaken over a wide area. Victoria has been the main center of activity. For the ten months ended Oct. 31, 89 companies, with an aggregate subscribed capital of £1,731,550, have been registered in that State, compared with 39 companies, with a capital of £674,820, for the whole of 1931. Companies formed during recent weeks include Maldon Gold Development, to unwater and further develop the New Chum and South German mines at Maldon, Victoria; New Chum Syndicate Gold, to open a mine at Bendigo, Victoria; Guildford Plateau Gold Mines, to operate at Guildford, Victoria; Golden Slipper Mines, to develop a property at Glenluce, Victoria. White Hope South, to acquire leases at Cracow, Queensland. . . . An English company, Goldfields Australian Development, has been formed by Consolidated Goldfields of South Africa, which company holds large interests in several Australian companies, including Wiluna Gold, Gold Mines of Australia and Lake View & Star, in conjunction with the Transvaal Agency and important Broken Hill interests. The first work to be undertaken is the investigation of low-grade properties at Wiluna and Laverton, Western Australia. . . . The consulting engineers of Gold Mines of Australia, C. C. Gibson and G. L. Clark, have reported unfavorably upon the Granites gold field in Central Australia. Granites West, recently formed with a capital of £200,000, will probably cease operations. Gold Mines of Australia has a substantial interest in Granites West. A second company at work on the field, Chapmans Gold Mines, intends to continue operations and is sinking wells to obtain water, lack of which is making living conditions on the field almost unbearable. Two small test batteries will be erected.

Large Kalgoorlie Plant Projected. Great Boulder Proprietary, Kalgoorlie, has arranged with Charles Ruwolt Proprietary, Ltd., of Richmond, Victoria, the largest company manufacturing mining equipment in Australia, to undertake the design and construction of a new crushing and flotation plant with large capacity. The work of installation will be initiated early in 1933; meanwhile, underground development at the Great Boulder is being speeded up. . . . Broken Hill Proprietary has exercised its option over the Hannan's North Extended mine, at the north end of the Kalgoorlie field. . . . The new 250-ton bromo-cyanide plant recently placed into commission on the Boulder Perseverance, Kalgoorlie, is giving a recovery of 93 per cent, with markedly decreased costs. The old treatment plant will be dismantled. . . . Extensions to the diesel power plant on the Lake View & Star are well in hand. The new three-compartment shaft on the Chaffers lease, to which the ore from the mine workings will be hauled by electric locomotives, has been completed to the 1,300-ft. level. On the Lake View lease at the 1,400-ft. level a horizontal drill hole in hitherto unprospected ground penetrated a lode channel 68 ft. wide at 334 ft. from the level drive. Grade was low, but 30 in. assayed 54/- per ton. The find is regarded as of major importance, as oreshoots are likely to occur along this well-defined ore channel.

Plant Developments in Queensland. At Mount Isa, Queensland, three large May-type plunger jigs have been installed and are giving a lead concentrate equal in grade to that obtained in the flotation plant. During October when the record of 60,149 tons of crude ore was milled, the jigs produced 746 tons assaying 46.9 per cent lead and 16.7 oz. silver; and the flotation section, 11,533 tons assaying 46.5 per cent lead and 18.8 oz. silver. The jigs will ease the load on the flotation section and improve recoveries. The coke-oven plant being erected by the Queensland Government at Bowen will be in commission by February and will supply coke for Mount Isa smelters at a price substantially less than at the present time. . . . Satisfactory results are being obtained at Mount Morgan, Queensland, where a test plant has been handling 100 tons of ore daily, giving a 5-oz. concentrate with a 75 per cent recovery. Additional fine grinding, classifying, and filtering equipment is being installed, to permit 150 tons to be treated daily. To treat the required 3,000 tons of ore weekly, an expenditure of £30,000 will be necessary. The company does not expect difficulty in obtaining this amount. . . . Mount Coolon Gold Mines, Queensland, had the first clean-up at the beginning of November, when 1,135 oz. gold was obtained from 1,600 tons of ore treated experimentally in starting up the new 200-ton mill. A second clean-up at the middle of November yielded 1,116 oz. fine gold from 1,590 tons, equivalent to 14 dwt. per ton. Absorption of gold in various portions of the mill reduced the yield considerably. The company is now carrying out vigorous development and is sinking the main shaft below the 300-ft. level.

Progress in New Guinea. Alluvial Gold, a gold-dredging company recently formed by Alluvial Tin (Australia), has agreed to bore an extensive gold-dredging property owned by Bululo Gold Deposits on the New Guinea gold fields. Should boring presage profitable operations, a company with a capital of £1,000,000 will be formed to work the area with large electric dredges. . . . Bululo Gold's second dredge in the Bululo area in New Guinea commenced work on Oct. 25. The two dredges for the Bulowat area are to dig deeper and will have buckets of larger capacity. No. 3 dredge is expected to be in operation in August, 1933, and No. 4 in February, 1934. For the four weeks' period ended Nov. 7, the Bululo company's two dredges treated 268,200 cu.yd. for 5,371 oz. bullion, worth $72,508 (gold). ■

Nuhana flotation mill at Ehimeken, Japan. Rice fields and tailing dump in foreground.

Daily and Average Monthly Prices of Metals

December, 1932

United States Market

Dec.	Electrolytic Copper: Domestic Refinery	Electrolytic Copper: Export Refinery	Straits Tin New York	Lead: New York	Lead: St. Louis	Zinc St. Louis
1	4.900@5.025	5.025	22.650	3.000	2.875@2.900	3.125
2	4.900@5.025	5.000	22.425	3.000	2.875@2.900	3.125
3	4.900@5.025	4.950	22.325	3.000	2.875@2.900	3.125
5	4.900@5.025	4.950	22.450	3.000	2.875@2.900	3.125
6	4.775@5.025	4.875	22.500	3.000	2.875@2.900	3.125
7	4.775@5.025	4.825	22.700	3.000	2.875	3.125
8	4.775	4.800	22.650	3.000	2.875	3.125
9	4.775	4.700	22.550	3.000	2.875	3.125
10	4.775	4.650	22.600	3.000	2.875	3.125
12	4.775	4.600	22.650	3.000	2.875	3.125
13	4.775	4.575	22.550	3.000	2.875	3.125
14	4.775	4.550	22.550	3.000	2.875	3.125
15	4.775	4.575	22.800	3.000	2.875	3.125
16	4.775	4.600	22.800	3.000	2.875	3.125
17	4.775	4.650	22.750	3.000	2.875	3.125
19	4.775	4.700	22.850	3.000	2.875	3.125
20	4.775	4.825	22.850	3.000	2.875	3.125
21	4.775	4.800	22.850	3.000	2.875	3.125
22	4.775	4.725	22.850	3.000	2.875	3.125
23	4.775	4.750	22.800	3.000	2.875	3.125
24	4.775	4.750	22.750	3.000	2.875	3.125
26	Holiday					
27	4.775	4.750	22.700	3.000	2.875	3.090@3.125
28	4.775	4.700	22.750	3.000	2.875	3.125
29	4.775	4.775	22.750	3.000	2.875	3.125
30	4.775	4.800	22.750	3.000	2.875	3.125
31	4.775	4.825	22.750	3.000	2.875	3.125
Av. for Month	4.813	4.759	22.677	3.000	2.877	3.124
Dec.			**Averages for Week**			
7	4.942	4.938	22.508	3.000	2.885	3.125
14	4.775	4.646	22.592	3.000	2.875	3.125
21	4.775	4.800	22.850	3.000	2.875	3.125
28	4.775	4.735	22.770	3.000	2.875	3.122
Dec.			**Calendar Week Averages**			
3	4.990	5.004	22.404	3.000	2.888	3.129
10	4.848	4.800	22.575	3.000	2.879	3.125
17	4.775	4.592	22.683	3.000	2.875	3.125
24	4.775	4.758	22.825	3.000	2.875	3.125
31	4.775	4.770	22.740	3.000	2.875	3.122

Silver, Gold, and Sterling Exchange New York and London

Dec.	Sterling Exchange: "Checks"	Sterling Exchange: "90-Day Demand"	Silver: New York	Silver: London	Gold London
1	3.211250	3.215000	25.250	17.5625	127s 8½d
2	3.207500	3.210000	25.000	17.5000	127s 3d
3	3.186250	3.190000	24.875	17.5000	.....
5	3.178750	3.181250	25.125	17.5625	130s ½d
6	3.200000	3.202500	25.250	17.6875	129s 2d
7	3.223750	3.228750	25.500	17.7500	128s 1d
8	3.225000	3.228750	25.500	17.6250	126s 8d
9	3.233750	3.237500	25.375	17.6250	127s 11d
10	3.260000	3.265000	25.500	17.5625	.....
12	3.260000	3.265000	25.375	17.5000	126s 2d
13	3.260000	3.263750	25.125	17.3125	126s 5d
14	3.292500	3.297500	25.125	17.1250	126s 6d
15	3.286250	3.291250	25.125	17.0625	125s 3½d
16	3.301250	3.305000	25.125	17.0000	124s 9½d
17	3.310000	3.313750	25.125	17.0000	.....
19	3.315000	3.320000	25.250	17.0000	124s 1½d
20	3.325000	3.328750	25.250	16.9375	123s 8d
21	3.335000	3.340000	25.000	16.6875	123s 5d
22	3.326250	3.330000	24.875	16.5625	123s 4½d
23	3.330000	3.333750	24.875	16.8125	123s 11d
24	3.326250	3.331250	24.625	16.5625	.....
26			Holiday		
27	3.332500	3.336250	24.625	Holiday	
28	3.328750	3.333750	24.375	16.5000	123s 8½d
29	3.312500	3.316250	24.250	16.2750	123s 11d
30	3.303750	3.306250	24.375	16.4375	124s 5d
31	3.326250	3.331250	24.375	16.5000	.....
Av. for Month	327.683	.....	25.010	17.110	.....
Dec.		**Averages for Week**			
7	3.20125	.....	25.167	.....	.....
14	3.25521	.....	25.333	.....	.....
21	3.31208	.....	25.146	.....	.....
28	3.32875	.....	24.675	.....	.....

Calendar week averages: New York Silver, Dec. 3rd, 25.271; 10th, 25.375, 17th, 25.167; 24th, 24.979; 31st, 24.400.

London Market

December	Copper: Standard Spot	Copper: Standard 3 Mo.	Copper: Electrolytic (Bid)	Tin: Spot	Tin: 3 Mo.	Lead: Spot	Lead: 3 Mo.	Zinc: Spot	Zinc: 3 Mo.
1	31.5000	31.8125	36.75	151.500	152.875	11.7500	12.1875	15.0625	15.3750
2	31.6250	32.0000	36.50	151.750	153.125	11.9375	12.3750	15.3125	15.6250
5	31.0000	31.3750	36.50	150.750	151.875	11.4375	11.8125	15.1875	15.5000
6	30.7500	31.0625	36.00	151.875	152.500	11.3125	11.7500	15.1250	15.4375
7	30.7500	31.1250	35.75	151.625	152.625	11.3125	11.7500	15.1250	15.4375
8	30.0625	30.4375	35.00	151.875	152.625	11.2500	11.6250	15.0625	15.3125
9	29.1875	29.5625	34.50	150.250	151.375	11.0000	11.3750	14.8750	15.1250
12	27.2500	27.6250	32.50	149.125	150.250	10.8750	11.3125	14.7500	14.9375
13	28.0000	28.3750	33.50	149.625	150.750	11.0000	11.4375	15.0625	15.1875
14	27.7500	28.0625	33.00	148.250	149.250	10.9375	11.3750	15.0625	15.1875
15	27.7500	28.1250	33.00	149.125	150.375	11.0000	11.3750	15.3750	15.3750
16	28.0000	28.1875	32.75	149.625	150.750	11.1250	11.5000	15.3750	15.4375
19	29.0625	29.4375	34.25	149.375	150.750	11.3750	11.8125	15.5625	15.6250
20	28.6875	29.0000	33.50	149.000	150.500	11.1875	11.5625	15.4375	15.4375
21	28.1875	28.5625	33.75	148.750	150.125	11.0625	11.4375	15.3125	15.3125
22	28.0625	28.3750	33.50	148.500	149.875	11.0000	11.3750	15.3750	15.3750
23	28.1250	28.4375	33.75	148.625	149.875	10.8125	11.3125	15.3750	15.3750
26	Holiday								
27	Holiday								
28	28.5000	28.8750	33.75	148.375	149.750	10.7500	11.1875	15.4375	15.4375
29	28.8750	29.2500	34.375	149.125	150.250	10.7500	11.1250	15.3125	15.3750
30	28.6250	28.9375	34.25	149.000	150.000	11.0000	11.1250	15.0000	15.0625
Average for month	29.088	.....	34.344	149.806	.....	11.144	11.541	15.209	15.347

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are 1c. above St. Louis

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1931	1932	1931	1932	1931	1932
January	29.423	29.780	13.810	19.623	485.260	342.515
February	26.773	30.136	12.432	19.573	485.551	345.141
March	29.192	29.810	13.524	18.336	485.596	363.463
April	28.279	28.298	13.120	16.923	485.764	374.731
May	27.650	27.755	12.858	16.868	486.188	367.370
June	27.250	27.466	12.707	16.844	486.291	364.471
July	28.255	26.700	13.197	16.930	485.349	354.865
August	27.524	27.986	12.815	18.000	485.284	347.449
September	28.180	27.870	14.101	17.998	451.245	347.003
October	29.538	27.195	17.153	17.813	388.029	339.500
November	32.223	26.698	19.393	18.099	371.130	327.458
December	30.120	25.010	20.023	17.110	336.798	327.683
Year	28.700	27.892	14.594	17.843	452.707	350.137

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery Electrolytic		London Spot Standard		London Spot Electrolytic	
	1931	1932	1931	1932	1931	1932
January	9.838	7.060	44.938	39.459	47.524	46.200
February	9.724	5.965	45.372	36.917	47.950	41.381
March	*9.854	5.763	44.818	33.039	47.699	36.786
April	9.392	5.565	42.694	29.943	45.375	34.190
May	8.665	5.237	38.897	28.548	42.175	32.833
June	8.025	5.145	35.827	26.872	38.966	30.841
July	7.698	5.053	34.402	26.071	37.293	29.107
August	7.292	5.219	32.572	31.401	35.388	34.784
September	6.988	5.978	31.503	34.986	36.148	38.318
October	6.775	5.733	34.957	31.890	41.000	36.190
November	6.558	5.131	35.854	31.972	41.190	36.568
December	6.580	4.813	38.273	29.088	44.409	34.344
Year	8.116	5.555	38.342	31.682	42.093	35.962

New York quotations, cents per pound. London, pounds sterling per long ton. Export copper, f.o.b. refinery: Average for December, 4.759

Lead

	New York		St. Louis		London			
	1931	1932	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.802	3.750	4.604	3.550	13.872	13.905	15.084	15.128
February	4.552	3.712	4.340	3.499	13.444	13.550	14.560	14.571
March	4.527	3.150	4.276	2.993	13.128	13.355	12.345	12.634
April	4.412	3.000	4.164	2.900	12.375	12.606	11.223	11.503
May	3.818	3.000	3.651	2.900	11.491	11.778	10.673	11.036
June	3.917	2.993	3.761	2.896	11.582	11.952	9.608	9.898
July	4.400	2.747	4.225	2.611	12.731	12.899	9.818	10.152
August	4.400	3.235	4.225	3.093	11.944	11.994	11.349	11.588
September	4.400	3.465	4.217	3.315	11.932	12.026	13.122	13.349
October	3.964	3.052	3.773	2.939	13.227	13.270	11.958	12.170
November	3.937	3.050	3.758	2.931	14.577	14.491	12.071	12.395
December	3.792	3.000	3.592	2.877	15.188	15.361	11.144	11.541
Year	4.243	3.180	4.049	3.042	12.958	13.099	11.913	12.164

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1931	1932	1931	1932
	Straits		Spot	
January	26.137	21.804	115.798	140.219
February	26.315	22.018	117.919	139.143
March	27.065	21.863	121.852	129.810
April	25.222	19.244	112.775	108.935
May	23.221	20.948	104.331	122.286
June	23.478	19.659	104.966	114.530
July	24.978	20.931	111.478	125.863
August	25.738	22.985	114.875	142.017
September	24.618	24.779	117.813	152.705
October	22.723	23.936	126.932	151.280
November	22.779	23.354	132.857	153.585
December	21.328	22.677	138.909	149.806
Year	24.467	22.017	118.375	135.848

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.035	3.011	12.747	13.113	14.416	14.834
February	4.012	2.817	12.303	12.694	13.872	14.289
March	4.002	2.787	12.190	12.676	12.616	13.024
April	3.717	2.725	11.353	11.838	11.670	11.958
May	3.306	2.532	10.484	10.875	12.432	12.682
June	3.416	2.777	11.270	11.750	11.548	11.866
July	3.893	2.537	12.280	12.802	11.592	11.967
August	3.917	2.758	11.444	12.028	13.594	13.844
September	3.744	3.322	11.571	12.063	15.455	15.616
October	3.377	3.027	12.733	13.216	14.869	15.140
November	3.209	3.094	13.845	14.247	15.264	15.534
December	3.149	3.124	14.361	14.818	15.209	15.347
Year	3.640	2.876	12.215	12.677	13.545	13.842

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1931	1932	1931	1932
January	67.115	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	55.000	23.300	23.300
April	55.000	55.000	23.300	23.300
May	55.000	55.000	23.300	23.300
June	55.000	55.000	23.300	23.300
July	55.000	55.000	23.300	23.300
August	55.000	55.000	23.300	23.300
September	55.000	55.000	23.300	23.300
October	55.000	55.000	23.300	23.300
November	55.000	55.000	23.300	23.300
December	55.000	55.000	23.300	23.300
Year	56.010	55.000	23.300	23.300

Aluminum in cents per pound, 99 per cent grade. Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1931	1932	1931	1932	1931	1932
January	7.317	5.976	103.000	64.900	36.000	40.000
February	7.069	6.489	100.205	66.304	34.000	40.000
March	7.127	6.188	100.423	72.537	29.538	40.000
April	6.888	5.746	102.077	72.125	26.346	40.000
May	6.524	5.170	101.140	66.380	24.980	39.500
June	6.342	5.034	92.058	59.481	37.115	37.500
July	6.802	5.000	85.808	53.580	40.000	35.200
August	6.596	5.144	80.115	47.444	40.000	35.000
September	6.542	5.606	76.300	47.500	40.000	33.800
October	6.517	5.575	72.385	47.600	40.000	33.000
November	6.679	5.771	68.587	48.750	40.000	33.000
December	6.231	5.400	66.115	48.500	40.000	30.462
Year	6.720	5.592	87.351	57.925	35.665	36.455

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Pig Iron[1]

	Bessemer		Basic		No. 2 Foundry	
	1931	1932	1931	1932	1931	1932
January	17.50	16.00	17.00	15.00	17.00	15.50
February	17.29	15.68	16.79	14.68	16.79	15.18
March	17.00	15.50	16.50	14.50	16.50	15.00
April	17.00	15.50	16.50	14.50	16.50	15.00
May	17.00	15.50	16.38	14.50	16.44	15.00
June	17.00	14.85	15.50	14.35	16.00	14.75
July	17.00	14.56	15.50	14.00	16.00	14.50
August	17.00	14.50	15.50	14.00	16.00	14.50
September	17.00	14.50	15.50	14.00	16.00	14.50
October	16.92	14.50	15.42	14.00	16.00	14.50
November	16.50	14.50	15.00	14.00	16.00	14.50
December	16.25	14.50	15.00	14.00	15.75	14.50
Year	16.96	15.01	15.88	14.29	16.25	14.79

Iron in dollars per long ton. [1]F.o.b. Mahoning and Shenango Valley furnaces; freight to Pittsburgh, $1.76.

prices for the Prime Western grade.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent forenoon market demand. Cables command a premium.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft.

All Figures Except Tin Represent Tons of 2,000 Lb.

	1931	1932									
COPPER — North and South America	Dec.	Jan.	Feb.	March	April	May	June	July	Aug.	Sept.	Oct.	Nov.
Production, refined, daily average	2,780	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Domestic shipments	36,972	.....	.....	.....	.....	.....	.....	.....	(a) 18,300	(a) 19,700	(a) 25,500	(a) 25,000
Foreign shipments	26,032	.....	.....	.....	.....	.....	.....	.....	(a) 48,000	(a) 53,700	(a) 55,500	(a) 55,000
Stocks, blister and refined	722,014	.....	.....	.....	.....	.....	.....	.....	.....	(b) 775,000	(b) 773,000	(b) 770,000
LEAD — United States												
Production, refined, from domestic ore	33,576	32,180	28,081	30,345	23,236	25,902	26,068	15,819	17,118	20,498	21,092	24,465
Production, secondary and foreign	4,031	3,554	3,920	3,989	3,134	2,941	2,033	3,754	6,308	2,315	6,984	2,873
Production, total, daily rate	1,213	1,185	1,103	1,107	879	930	936	631	756	760	905	901
Shipments reported	30,297	27,867	26,319	31,162	26,081	24,258	21,511	19,723	28,248	26,413	29,764	22,838
Stocks, end of month	151,380	160,257	165,933	169,091	169,091	173,929	180,460	180,255	175,426	171,831	170,171	174,629
ZINC — United States												
Production, daily average	709	723	742	726	688	601	547	476	432	434	480	525
Domestic shipments	21,005	22,472	21,896	22,576	18,046	18,087	14,958	12,896	18,069	21,132	18,801	15,674
Export shipments	.....	31	.....	.....	.....	.....	24	.....	39	20	20	20
Stocks, end of month	129,825	129,886	129,506	129,451	132,025	132,580	134,032	135,907	131,203	123,056	119,101	119,159
World Production Rate (Daily Average)												
Copper	3,940	.....	.....	.....	.....	.....	.....	.....	.....	(b) 2,500	(b) 2,477	(b) 2,480
Lead	3,976	3,938	3,872	3,598	3,587	3,711	3,497	3,190	3,334	3,323	3,510	3,518
Zinc	2,601	2,600	2,615	2,623	2,528	2,392	2,343	2,224	2,121	2,120	2,244	2,336
Tin (Metric Tons)	351	311	274	340	312	311	261	178	179	255	247	.....

(a) Deliveries from all sources; private estimate. (b) *E.&M.J.* estimate.

Editor: A. W. Allen

Associate Editors: A. H. Hubbell, H. D. Keiser

Assistant Editor W. N. P. Reed

Market Editor: H. H. Wanders

Publishing Director: H. W. Clarke

Engineering and Mining Journal

Contents for January, 1933

Copyright 1933, by McGraw-Hill Publishing Company, Inc.

Volume 134 Number 1

Editorial 1

Large Crushing and Screening Plant ..... *L. A. Slattery* 2

Low Power Costs from Diesel-Electric Plant ..... *O. J. de Gaetano* 12

Microscopical Determination ..... *A. S. Hecht* 14

Cost of Surface Metal Mining ..... *M. J. Elsing* 17

Mining and Furnacing Quicksilver Ore ..... *N. H. Stearn* 22

Boltless Cylindrical Mill Lining ..... *E. D. McIntyre* 25

Copper Matte Composition—II ..... *C. K. Avetisian* 27

Level Development ..... *Lucien Eaton* 29

Book Reviews 34

Personal Notes, Comment, Criticism 36

Industrial Progress 38

Trends and Developments in the Industry 41

Market Data 46

McGRAW-HILL PUBLISHING COMPANY, INC.
330 West 42d St., New York, N. Y.
Cable Address: McGrawhill, N. Y.

McGRAW-HILL PUBLISHING COMPANY, LTD.
Aldwych House, Aldwych, London, W.C. 2, England
Cable Address: Backlash, Estrand, London

James H. McGraw, Chairman of the Board; Malcolm Muir, President; James H. McGraw, Jr., Vice-President and Treasurer; Mason Britton, Vice-President; Edgar Kobak, Vice-President; H. C. Parmelee, Vice-President and Editorial Director; Harold W. McGraw, Vice-President; C. H. Thompson, Secretary.

Complete Market News and Prices

THE monthly magazine you are now reading carries only a condensed statistical summary of prices of the major metals. For buyers and sellers of ores, metals, minerals, and scrap who require reliable information as soon as it is available, we now provide *Metal and Mineral Markets*, which goes to press at the close of the metal-market week each Wednesday. It is printed at high speed and is in the mails that evening. In compact form, ready for insertion in a ring binder, it is ideal for desk and reference use. It carries the standard *E.&M.J.* quotations used in contracts the world over, recognized as authoritative and dependable. *Metal and Mineral Markets* is priced to subscribers in the United States and its possessions at $3 per annum; to countries in the Americas but outside the United States, $6; elsewhere, $10 yearly: 52 issues. Address *M.&M.M.*, 330 West 42d St., New York City.